

HANDBOOK OF NONFERROUS METALLURGY

PRINCIPLES AND PROCESSES

HANDBOOK OF NONFERROUS METALLURGY

Edited by

DONALD M. LIDDELL

- I. PRINCIPLES AND PROCESSES
- II. RECOVERY OF THE METALS

Handbook of Nonferrous Metallurgy

PRINCIPLES AND PROCESSES

PREPARED BY A STAFF OF SPECIALISTS

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HANDBOOK OF NONFERROUS METALLURGY
PRINCIPLES AND PROCESSES

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IV

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PREFACE OF THE SECOND EDITION

TH E improvements in the art of metallurgy have rendered a new edition of this handbook necessary, although it is to be regretted that certain information known to the contributors and to the Editor must remain unpublished because of security considerations.

In general, the plan of the first edition is repeated in the second. An attempt has been made to conform more closely to the idea of treating only of the reduction and refining of the metals and in the main to omit considerations of their working and utilization after production. This has resulted in the deletion of the chapter on Metallography and the addition of a chapter on Drying in its place.

Another departure from the general plan of the first edition is that each volume is indexed separately and it will be necessary to consult the indexes of both in order to make certain one has all the references to the subjects.

I wish to make special acknowledgement of the careful reading given the chapter on Aluminum by Mr. G. C. Riddell.

DONALD M. LIDDELL.

NEW YORK, N.Y.,
August, 1945.

PREFACE OF THE FIRST EDITION

THERE are certain processes and materials that are common to all metallurgical operations: the use of refractories, of fuels, general methods of concentration, of filtration, etc. It has been my attempt to pick out these metallurgical common factors and to give each of them treatment in a separate chapter, following these chapters with others describing the metallurgy of each metal; or, in some cases, two or more metals, when there is no basic difference in their metallurgy, are treated in one chapter.

The scheme of treatment in the various chapters is not uniform—it intentionally is not so. In general the attempt is made to treat at greatest length those metals of prime commercial importance, but this plan is departed from if it is felt that existing literature on a certain metal is not readily accessible to the ordinary metallurgical engineer. While with copper, lead and similar metals, the subject matter of the chapter is the extraction of the metal from the ore, in the case of such metals as magnesium and aluminum, where it is extremely unlikely that the average engineer will ever be called upon to design a reduction works, or to work in any plant except one with an established metallurgy, much of the space is given to the useful alloys of the metal and how to work with them. In some cases analytical methods are given when it is felt that existing literature is deficient.

There is much that is elementary. The book is designed for the student as well as for the engineer and consultant, yet it is hoped that there is enough advanced material to make the book useful to the metallurgist of experience, particularly if he is confronted with problems somewhat outside his own specialty. There is no attempt to give extensive tables of basic data—for these the seeker is referred to International Critical Tables, the tables of the International Congress of Applied Chemistry, or to Landolt, or to the editor's "Metallurgists' and Chemists' Handbook."

To the contributors is due more than their signed work indicates, as there has been much cooperative correction and criticism. I must, however, acknowledge special indebtedness to Percy E. Barbour, who took over all work on the book during my various absences from the United States during the period of its preparation, and also to H. A. Megraw and Dr. Colin G. Fink, who have given liberally of their time, and to Thomas A. Wright, of Lucius Pitkin & Co., for his hints regarding rare-metal recovery.

DONALD M. LIDDELL. -

BALTIMORE, MD.,
May, 1926.

CONTENTS

	PAGE
PREFACE TO THE SECOND EDITION.	vii
PREFACE TO THE FIRST EDITION.	ix
CHAPTER	
I. INTRODUCTION (Liddell).	1
II. DRYING (Russell)	6
III. CRUSHING AND GRINDING (Hardinge).	18
IV. SAMPLING (Liddell).	60
V. SCREENING (Reed).	89
VI. PART I. CLASSIFICATION (Dorr, Bosqui).	101
PART II. CONCENTRATION (Liddell).	126
PART III. FLOTATION (Swainson)	149
VII. PART I. DEWATERING (Dorr, Bosqui).	167
PART II. FILTRATION (Irvin)	186
VIII. METALLURGICAL FUELS (Liddell).	214
IX. PYROMETRY (Roeser, Van Dusen)	230
X. ROASTING AND SINTERING (Dwight, Rowen)	276
XI. REFRACTORIES (Harvey)	324
XII. SEPARATION OF SOLIDS AND LIQUIDS FROM GASES (Landolt, Viets)	395
XIII. BRIQUETTING APPLIED TO METALLURGY (Stillman, Liddell)	409
XIV. POWER PLANT AND ACCESSORIES (Donaldson).	429
XV. MATERIALS OF METALLURGICAL CONSTRUCTION (Church).	456
XVI. PLANT LAYOUT (Barbour).	575
XVII. THE ELECTRIC FURNACE IN NONFERROUS METALLURGY (Liddell)	583
XVIII. APPLIED ELECTROCHEMISTRY (Fink)	629
INDEX.	647

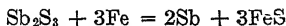
HANDBOOK OF NONFERROUS METALLURGY PRINCIPLES AND PROCESSES

CHAPTER I INTRODUCTION

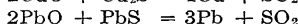
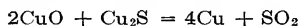
What Is a Metal?—The ordinary description of a metal is based on the appearance which is common to practically all metals—an appearance so individual that certain lexicographers use the term “metallic luster” in defining the word “metal.” Ordinarily, metals are good conductors of heat and electricity, and most of them are malleable and ductile. A metal is better defined as an element that will replace the hydrogen of acids, or, better still, as an element, but not including hydrogen, that takes a positive charge when a compound containing it is dissociated in solution. This definition rules out “the hypothetical metal, ammonium,” of the older chemistries. On the other hand, it allows for the amphoteric character of some metals, their tendency to form both acids and bases at different times, since by this definition an element that at any time takes a positive charge is a metal, and no element thus classified will, except it be combined in a complex ion, take a negative charge. That is, aluminum is a metal, for the simple ion is Al^{+++} , even though it is possible to have AlO_2^- . Platinum is a metal, for Pt^{++} or Pt^{+} exists, although in a complex ion PtCl_6^- occurs.

Metals occur in nature chiefly as oxides and sulphides, also very often as carbonates, silicates, and in their native forms, but at times they are combined with all the nonmetals. Because of the diverse natures of the compounds and the peculiarities of the metals themselves, there can be no uniform method of extraction—in short, the extraction of each metal is itself a study and demands a separate chapter in a work on general metallurgy. In general, however, the processes of extraction are: (1) the reduction of oxides, (a) most commonly by carbon, but often (b) by other metals; (2) the roasting of sulphides to oxides, then reduction; (3) the decomposition of salts by other metals; (4) electrolysis, both in aqueous and fused electrolytes; (5) double decompositions; (6) chemical processes not included in any of the above.

The greatest tonnage of metals is extracted by method 1a, since all iron and steel is so produced, and much lead and zinc, and formerly much copper. Method 1b is commercially important today in the thermit reductions by aluminum, and formerly produced all the aluminum supply. Much of the lead and zinc supply is produced by method 2. Antimony and bismuth are produced by method 3.



The modern electrolytic processes for the production of zinc and copper following the leaching of the ores are examples of aqueous electrolysis; methods for the production of magnesium, sodium, calcium, and aluminum are examples of electrolysis in fused electrolytes; while the reaction process for the production of lead and the converting of copper mattes illustrates double decomposition.



Method 6, chemical processes, includes the leading method for the extraction of gold—the cyanide process—and the methods of extracting selenium and tellurium, many processes for extracting silver, etc.

Occasionally, a metal is chiefly, and sometimes only, used in combination—still another illustration of extraction by chemical processes; as for radium, thorium, the rare earths, etc.

Even in metals produced by pyro processes, there are many furnaces from which to choose. The blast furnace produces the greatest tonnage of metal, although in nonferrous lines the reverberatory furnace is probably the greatest tonnage producer; much of the zinc supply is produced by distillation, while tube and crucible liquation furnaces are used for antimony and bismuth, and crucible furnaces, without any running off of the contents, also for the same elements.

These processes, then, introduce accompanying problems of roasting and smelting, fuels, refractories, crushing and grinding, sampling, fume control, etc. These subjects are first treated as to their general principles in subsequent chapters of this book, and then treated again in detail as applied to specific metals.

Ductility and malleability have been mentioned as general properties of metals. The most ductile metals, in order of decreasing ductility, are W, Au, Ag, Pt, Fe, Ni, Cu, Al, Zn, Sn, Sb, although some authorities place Al fourth. One ounce of tungsten wire 0.0005 in. in diameter measures 12,490 yd. (Fink). The finest wires are produced by covering a wire of one material with a coating of another, drawing the duplex wire thus made, and then dissolving off the outer coating; *e.g.*, a platinum wire coated with silver is drawn as fine as possible and the silver dissolved off in nitric acid, resulting in the finest obtainable platinum wire.

In order of decreasing malleability the common metals run Au, Ag, Al, Cu, Sn, Pt, Pb, Zn, Fe, Ni, Co. The thinnest leaf obtainable in 1914 was Au, 0.000008 cm.; Al, 0.000020; Ag, 0.000021; Pt, 0.000025; Cu, 0.000034; Dutch metal, 0.00007 cm. (Kaye and Laby). A metal may be brittle when either hot or cold (the condition being known as hot-short or cold-short) and not the reverse, or it may be both cold- and hot-short and malleable at some intervening temperature. Zinc is hot-short, but malleable at 150°C. Gold and silver have been made in sheets so thin as to be translucent, gold by transmitted light being green; silver, blue. Selenium near its melting point can be worked into translucent sheets of a beautiful crimson, but selenium when cold is as brittle as glass.

Brittleness or Toughness.—Martens gives a formula

$$\text{Toughness} = \frac{\text{ultimate strength}}{\text{yield point}} \times \frac{\text{per cent elongation}}{100}$$

and arranges the metals in the following order: Pb, Pt, Fe, Al, Ni, Zn, Sn, Cu, Ag, Au. Martens also states that

$$\text{Plasticity} = \frac{\text{toughness}}{\text{yield point}} \times 1,000$$

which reduces to

$$\frac{\text{Ultimate strength}}{\text{Yield point}^2} \times 10 \times \text{per cent elongation}$$

He places the plasticity of the metals in this order: Fe, Pt, Ni, Al, Zn, Cu, Ag, Au, Pb, Sn, and Kurnakoff-Schemtschuschny adds K, Na, Pb, Tl, Sn, Bi, Cd, Zn, Sb,

Tenacity.—The order usually given for tenacity is W, steel, Ta, Ni, Co, Fe, Cu, Al, Au, Zn, Sn, Pb. Tungsten in thin wires is ordinarily given a tensile strength of 600,000 lb. per sq. in.; steel (special steel, thin wires), 450,000 lb.; Ta, 150,000; Ni, hard-drawn, 96,000; Co, 75,000; rolled iron, 55,000; Cu, 50,000 to 60,000 lb. In general, metals in the form of thin wire or sheets have a higher tensile strength per square centimeter than the same materials in larger section.

Young's Modulus.—The pressure or tension on a bar in dynes per square centimeter divided by the compression or elongation produced by said pressure or tension per centimeter of length is known as Young's modulus. It is also defined in English measure as the pressure or tension in pounds per square inch divided by the compression or elongation per inch.

The modulus of elasticity (of which Young's modulus is a special case) is the ratio of the increment of unit stress to increment of unit deformation within the elastic limit.

Hardness.—Hardness is the power of a metal to resist abrasion and, of course, decreases markedly with an increase of temperature. The following list based on Bottone's scale of hardness, diamond = 30,010,¹ roughly represents hardnesses at ordinary temperatures:

Mn.	1456	Pd.....	1200	Au.....	979	Pb.....	570
Co..	1450	Pt.....	1107	Al.....	821	Tl.....	565
Ni..	1410	Zn.....	1077	Cd.....	760	Ca.....	405
Fe.....	1375	Ag.....	990	Mg.....	726	Na..	400
Cu.....	1360	Ir.....	984	Sn.....	651	K..	230

It seems probable that iridium is misplaced in the above table and should come above iron, but no definite data are available.

Fusibility.—The melting points range in order from those of gallium and mercury, which are liquid at ordinary temperatures, to those of platinum, iridium, tungsten, etc., which require the highest attainable artificial heats. For complete data see the atomic weight table in the Appendix. Small amounts of impurities usually lower the melting point of a metal. Many accepted melting points are probably low, owing to the difficulty of preparing pure metals to work on.

Heat and Electric Conductivity.—Most metals are good conductors of heat and electricity. Relative conductivities of some of the more common are given by Roberts-Austen as follows:

	Heat	Electricity
Au.....	100.0	71.0
Pt.....	98.4	9.77
Ag.....	97.3	100.0
Cu.....	87.8	99.5
Fe.....	37.4	12.4
Zn.....	36.0	25.9
Al.....	33.0	33.0
Sn.....	30.4	13.3
Pb.....	18.0	7.76

¹ Taken from ROBERTS-AUSTEN'S "Introduction to the Study of Metallurgy."

It is the popular belief that the conductivities for electricity and heat follow the same order, but the above table shows that such is not the case. The electrical resistivities of pure metals increase as the temperatures increase, and the conductivities of nonmetals decrease. Nevertheless, alloys of several metals may have a zero or even a negative coefficient of electrical resistivity for temperature increase. Among these may be noted Dr. Edward Weston's manganin, constantan, etc.

In similar manner it should be noted that, while all metals expand on heating, the alloys do not necessarily have a coefficient of thermal expansion which is the average of those of the component metals, for Invar has practically none.

The addition of small amounts of impurities lowers the electrical conductivity of any metal. The addition of 0.003 per cent of arsenic or of 0.008 per cent of antimony lowers the conductivity of copper by 1 per cent.

Hardness is usually increased by cold-working, either rolling or drawing, and decreased by annealing. But it must be noted that, while metal hardens rapidly during the initial stages of deformation, there is a stage, corresponding to a certain microstructure, beyond which the metal becomes softer, and in its final form the metal may be softer than in its original stage.

Magnetism.—While iron is the only element attracted strongly by a magnet, Ni, Co, Mn, Cr, Ti, Pd, Pt, and Os are also attracted somewhat, and the magnetic permeability of some of the platinum metals has led to some costly errors in cleaning up black-sand residues. (The metals in the above list are given in the order of decreasing permeability.) Some metals are repelled from magnets, *i.e.*, thin bars in a magnetic field tend to take a position at right angles to the lines of force. Arranged in the order of increasing repulsion, the metals are Bi, Sb, Zn, Sn, Cd, Hg, Pb, Ag, Cu, As, U, Ir, W. An alloy of Mn 24, Al 16, and Cu 60 per cent (Heusler's alloy) is attracted by a magnet, although its composition would not lead one to believe this would be the case. The alloys, or compounds, Fe₂B and MnB are strongly magnetic.

Metals may absorb gases in considerable volume. This is a constant factor in electrolytic work, where hydrogen is absorbed in the deposit. Nickel may absorb even up to 500 times its volume of hydrogen during deposition. Ordinarily, this hydrogen is given off under a light annealing. Silver strongly absorbs oxygen when molten, which is mostly given off as the silver solidifies, resulting in the familiar "sprouting" or "spitting" of cupel buttons. A good laboratory method for the determination of gases in metals is that of Simons, *Chem. & Met. Eng.*, Aug. 9, 1922, p. 249.

Fatigue of Metals.—Metals often fail under repeated stresses, although each stress is less than the elastic limit. C. E. Strohmeier of Manchester, England, about 1913 postulated that there was what he termed a "fatigue limit," a stress below which an infinite number of repeated stresses would not break the specimen and above which some finite number of stresses would cause breakage, this number decreasing as the stresses increased until it becomes unity at the ultimate tensile strength. He deduced that if the number of stresses at which the break takes place is N for a stress $\pm S_N$, there is a constant C and a fatigue limit F at which this relation holds true.

$$N + C' \sqrt[4]{\frac{10^6}{N}}$$

He determined the fatigue limits and constants for some alloys, the highest fatigue limit being 12.60 tons for chrome-nickel steel ($C = 0.50$) and ranging down to 2.16 tons for aluminum; 2.69 tons for annealed copper; 5.5 tons for rolled copper; 3.98 tons for cast iron; and 5.90 to 11.1 tons per sq. in. for mild steel, all values, it may be noted, well below the ultimate tensile strength.

The investigation has since been repeated by Prof. H. F. Moore of the University of Illinois. In this latter investigation, the point has been brought out that minute surface scratches and imperfections in the test piece apparently enormously decrease the fatigue limit.

All metals except bismuth and gallium expand on melting. Bismuth expands sharply on freezing, and this property is used in making type metal and other alloys in which sharp and delicate impressions are to be obtained by casting.

CHAPTER II

DRYING

By R. J. RUSSELL¹

Definition.—Broadly, the term “drying” means the process of removing a liquid (usually water) from solids, liquids, or gases. This includes absorption, adsorption, mechanical separation, and vaporization. This chapter is confined to drying defined as the removal of water from solids by vaporization, which is the most widely used process. However, economic considerations generally warrant removing as much water as possible by mechanical means before completing the drying by vaporization.

Drying is a necessary step in the processing of most raw materials to finished products. Frequently the subsidiary or contributing operations necessary to the processing also include a drying requirement. As an example, in the production of aluminum the principal material bauxite is dried, and the electrodes for the reduction cells are prepared from coke which requires drying.

Occasionally drying takes place without its recognition as a separate or specific step in a process. In some instances, ores and concentrates containing moisture are roasted, sintered, or smelted as a unit operation. Nevertheless, the work of removal of the mechanically held water is accomplished before any other physical or chemical change occurs.

Fundamentals of Drying.—The vaporizing of water is accompanied by the absorption of heat, and the rate of drying, within limits, is controlled by the rate heat is supplied. The heat for vaporization may be applied in three ways:

1. *Conduction*—Direct contact is maintained between a heated surface and the material being dried.

2. *Convection*—Air or other gas at elevated temperature is passed over or through the material.

3. *Radiation*—Radiant energy from a heated body is in part absorbed by the material and transformed into heat.

Frequently two and at times all three means for heat transmission are utilized in a drying operation. It is desirable to recognize each method as it is applied as well as the basic governing laws. However, practicable design involves correlation of the basic physical laws with empirical formulas developed from experimental work and observations of drying operations.

The second controlling factor in drying is the removal of vapor formed as the result of absorption of heat.

Drying proceeds (vaporization of water) as the vapor pressure of the water associated with a solid is raised above the vapor pressure of the surrounding atmosphere. Increasing the temperature raises the vapor pressure. Conversely, the vapor pressure of the surrounding atmosphere may be decreased to a level below that of the water. So long as a wetted surface is presented to the surrounding atmosphere, the drying rate is proportional to the difference between the vapor pressures.

When water is mechanically held in pores or interstices in the solid, the surface may become dry while water remains in the interior structure. Under these con-

¹ Hardinge Co., Inc., York, Pa.

ditions, the drying rate can proceed only as rapidly as the water proceeds by diffusion toward the surface of the structure.

The law of diffusion also enters into surface evaporation. The solid is surrounded by a film of gas, and water vapor from the solid surface diffuses through this relatively stationary film to the outer moving gas. As the relative velocity of the gas to solid is increased, the film thickness is decreased and surface evaporation is accelerated.

In practice, the velocity of the gas is limited to avoid carrying the solid from the drying chamber and the velocity of the solid is limited when fracture or breakage is detrimental. In some instances the material undergoing drying is conveyed by the

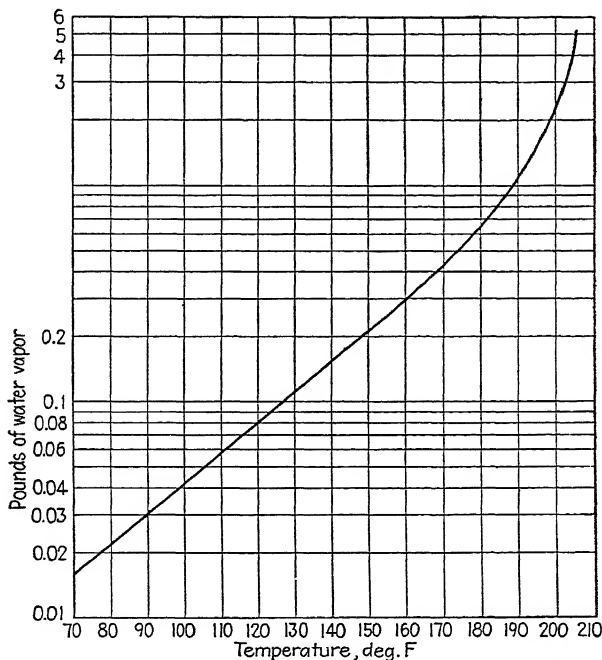


FIG. 1.—Saturation ratio, water vapor—air.

drying gases, and under these conditions means are provided to remove the dried solids from the spent drying gas.

The removal of the vaporized water from the drying chamber requires a gas-carrying medium. Exceptions are when the vapor pressure exceeds the pressure of the surrounding atmosphere (212°F. or higher at 29.92 in. of Hg) and when drying occurs in a partial vacuum.

A definite amount of water vapor can be carried by a given quantity of air or other gas depending on the temperature. When this limit is reached, the air or gas is saturated (100 per cent humidity). The slightest drop in temperature of the mixture results in condensation of part of the water vapor. Conversely, any increase in temperature permits an increase in the ratio of water vapor to dry gas. The amount of water vapor in a gas is the humidity of the gas. Although theoretically desirable,

in practice the air or gas discharging from a drying system is rarely saturated. The ratio of water vapor to dry air when saturated at various temperatures and standard barometric conditions is shown in Fig. 1.

From the foregoing, it is obvious that for any drying operation at normal atmospheric pressures, and where the exit gases are at a temperature below that corresponding to the boiling point of the water, there is a minimum quantity of air or other gas required for water-vapor removal. When the source of heat is by conduction or radiation, this minimum air requirement is always of great importance. When the heat required for drying is supplied by heated gases, the quantity of gas required for vaporization frequently exceeds the minimum limit.

Drying may be an intermittent (batch) or continuous process. When continuous, a fundamental consideration is the conveying of the material undergoing drying into, through, and from the drying chamber. The conveying must be controlled for proper time and contact of the material with the heat source and the vapor-removing medium. Continuous drying is preferred and in most instances is practicable.

Types of Driers.—Driers are classified as direct or indirect heat, depending upon the manner in which heat is supplied and utilized. *Direct-heat Driers.*—The material

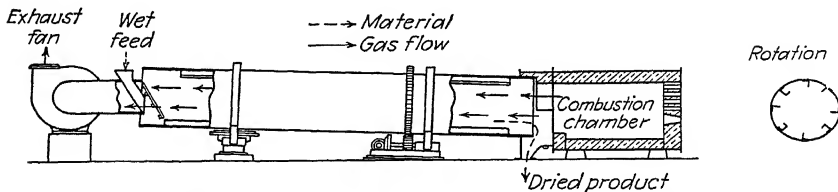


FIG. 2.—Ruggles-Coles single-shell direct-heat drier. (Hardinge Co., Inc.)

being dried is in direct contact with gases of combustion or heated air which provides the heat for drying. *Indirect-heat Driers.*—The material being dried is in contact with heated surfaces or subjected to radiant heat. The heated surfaces may be tubes, drums, cylinders, or other compartments heated by steam, hot gases, or liquids. Infrared lamps or other radiation media may also be a heat source. Both direct and indirect heating are used in some driers. These are classified as *semidirect* or *indirect-direct heat driers*.

Classes of Driers.—The means for mechanically handling the material undergoing drying may be used to classify types of driers.

Rotary Driers.—The rotary drier is a rotating cylindrical assembly with the end into which the undried material is introduced slightly elevated. Tires or riding rings are attached to the cylindrical shell, and support is provided by rollers or bearing wheels. Horizontal thrust wheels maintain the tires in position on the bearing wheels. A gear encircling the drier shell engages a driving pinion to provide rotation. Lifters and other internal attachments to the drier shell aid in passage of material through the drier and accelerate the drying process. A fan or stack exhausts the drying gases or air with the water vaporized during drying.

Sizes of rotary driers range from 18 to 144 in. in diameter with shell lengths from 10 to 150 ft. The shell and other parts in contact with the drying material and gases may be manufactured from steel, steel alloys, and nonferrous alloys. In some instances, refractory linings are used. Power requirements for rotating the shell range from 1 to 125 hp. and for the exhaust gas fan from 1 to 75 hp.

Rotary driers are adapted for direct and indirect heating and are further classified as follows:

Direct Heat.—1. A single-shell drier utilizing gases of combustion or heated air in direct contact with the material. The gases and material may travel in the same direction (parallel flow) or counter to each other (counterflow) (Fig. 2). The counter-flow drier is preferred when complete drying is necessary. Fine material carried out of the drier by the exhaust gases is not excessive, because the gases leaving the drier are in contact with the wet feed to the drier. The parallel-flow drier is preferred when the undried feed has a high initial moisture content and is of a sticky nature, but when some residual moisture in the dried product is acceptable. A fine material, completely dried in a parallel-flow drier, requires the use of efficient dust-collecting equipment to prevent excessive dust loss.

2. The hot gases from a manifold are directed into an open end of channels or louvers attached to the drier shell longitudinally and so arranged that the louvers overlap but provide openings the full length of the drier shell. The gases discharge through only the louver openings which are covered by material being dried. The gas flow through the drying mass is normal to the travel of material (Fig. 3).

In all direct-heat rotary driers, the spent drying gases transport the vaporized water removed during drying.

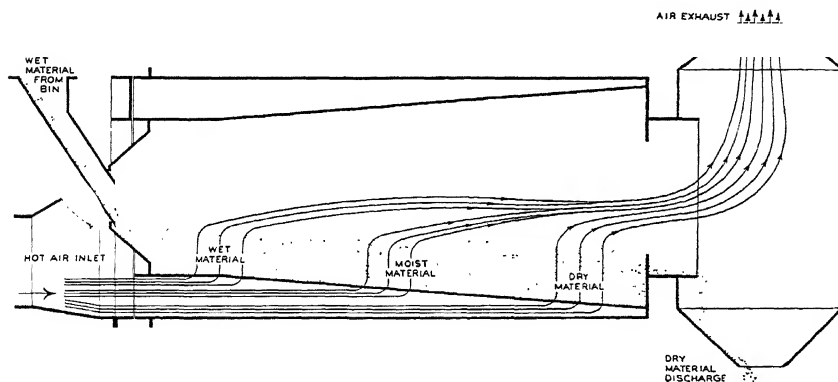


Fig. 3.—Direct-heat drier—circulation through louvers. (Link-Belt Co.)

Indirect Heat.—1. The exterior surface of the drier shell is heated by contact with gases of combustion in a chamber enclosing the shell through which the material is passed.

2. The heated gases pass through a central tube connected with a manifold at the discharge end of the drier, in which the gases are reversed and flow through ducts attached to the drier outer shell discharging from a gas exhaust head at the feed end of the drier. The material being dried travels along the annular space formed between the central flue and the outer shell with attached ducts. The material is heated by contact with, and radiation from, the central flue and the ducts (Fig. 4).

3. The heat source is a supply of steam to banks of tubes suspended longitudinally within the shell and connected to a steam header. Steam enters the header, and condensate is discharged by means of a dual rotary valve (Fig. 5).

For all types of indirect-heat driers, it is necessary to provide sufficient air flow to transport the vaporized water released from the drying material. For those types using combustion gases, a fan or stack is required to provide the required gas flow.

Indirect-direct or Semidirect Heat.—1. The gases of combustion pass around the exterior of the shell and then pass through the shell in contact with the material as in a direct-heat drier.

2. A double-shell drier is one in which the gases flow through the inner shell, reverse at the discharge end, and flow counter to the material passing through the annular space (Fig. 6).

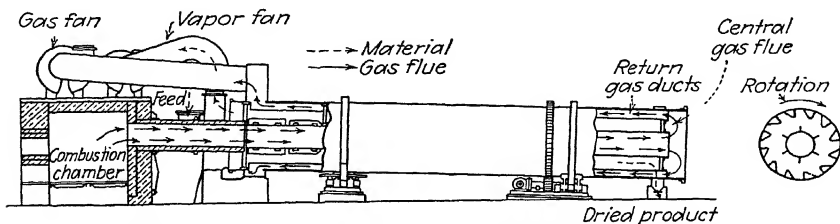


FIG. 4.—Ruggles-Coles indirect-heat drier hot-gas circulation. (Hardinge Co., Inc.)

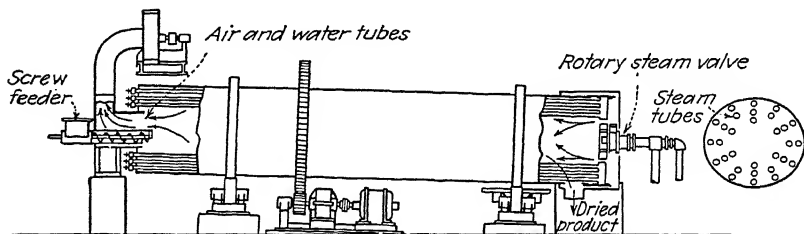


FIG. 5.—Indirect drying by steam. (Hardinge Co., Inc.)

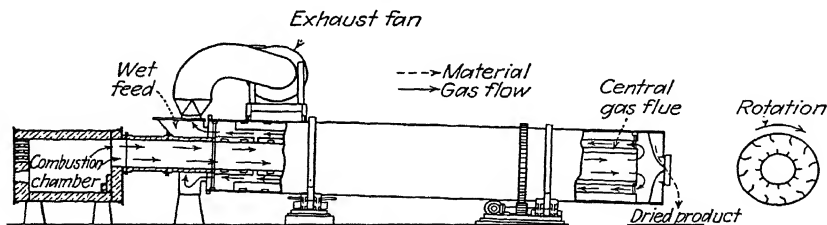


FIG. 6.—Ruggles-Coles double-shell semidirect-heat drier. (Hardinge Co., Inc.)

3. Combustion gases flow through ducts attached to the outer shell and return through the drier shell counter to the travel of material.

As in direct-heat driers, the gases that are the heat source also provide the carrying medium for the water vapor.

Conveyer-type Driers.—There are two principal types of conveyer driers.

1. *a.* The material to be dried is fed onto belts or screens that are mechanically driven and convey the material through a drying chamber. Direct-heat drying is practiced when heated air or other gas is caused to flow across the bed on the conveyer or through the bed when screens are used. Indirect-heat drying is accomplished by subjecting the mass traveling on the conveyer to radiant heat from infrared lamps or

other radiation media. The conveyer may consist of single or multiple belts. In some instances the undried material is partially dried by other means and extruded in strip or rod form so that it may be more readily dried with a conveyer type of drier.

b. The conveying means may be a screw or paddle-type conveyer rotating in a stationary housing externally heated by steam or combustion gases. Another type conveys the material along a horizontal deck by a reciprocating raking mechanism. The underside of the deck is heated by gases of combustion.

2. The means for conveying may be electric vibration. The direct-heat type has a perforated conveying surface through which the heated air or gas passes. The rate of travel and the depth of the drying bed are controlled by the rate of vibration which may be varied for individual drying sections. The drier may be composed of multiple tiers of sections (Fig. 7). The indirect-heat type has a solid plate conveying surface, and the material is heated by conduction through the conveying surface or by radiant heat.

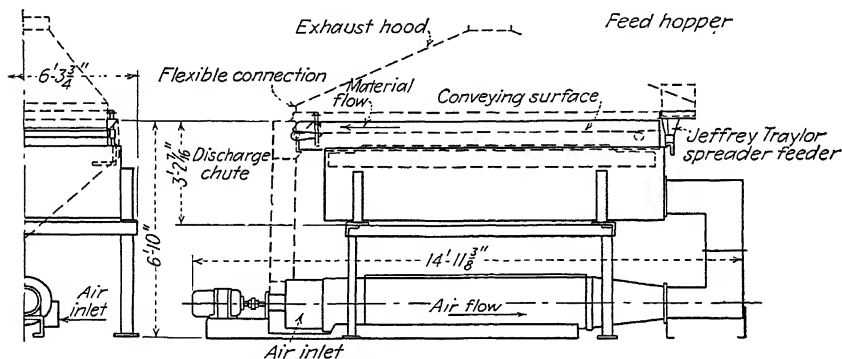


FIG. 7.—Vibrating conveyer-type drier. (Jeffrey Mfg. Co.)

The heated air or gas is the carrying medium for the removal of vaporized water for the direct-heat conveyer-type driers. With the indirect-heat driers a controlled air flow or natural convection current removes the vaporized water.

Drum Driers.—The material to be dried is fed as a slurry onto the outer surface of a rotating steam-heated drum. The drum speed is controlled so that the material is dried in less than one complete revolution of the drum. The drum may dip into the undried slurry, the feed may be discharged from a pipe or weir at the top of the drum, or the feed may be sprayed upon the drum. The thin film that clings to the heated drum is heated and dried through part of a revolution of the drum and when dry is scraped from the drum automatically by a knife or rotating brushes (Fig. 8).

Drum driers are of the single- or double-drum types. The principle of operation is the same for both types. Water vapor is removed by natural convection air currents or by controlled air circulation.

The drum drier may also be of the vacuum type wherein the rotating drum is enclosed and a partial vacuum maintained in the enclosure.

Rapid Vaporizing Driers.—In some instances, materials of high moisture content are dried very quickly. The types of driers for such drying operations are variously known as flash, spray, and dispersion driers. The basic drying principles are similar for the several types.

The material to be dried, usually of small average particle size and frequently having a high water to solids ratio, is introduced into the drying chamber or system in a manner that causes turbulence, violent agitation, or atomizing. In the drying chamber, the material is in contact with heated air or other gases. Drying is extremely rapid because of the tremendous exposed surface of the material. The turbulence or

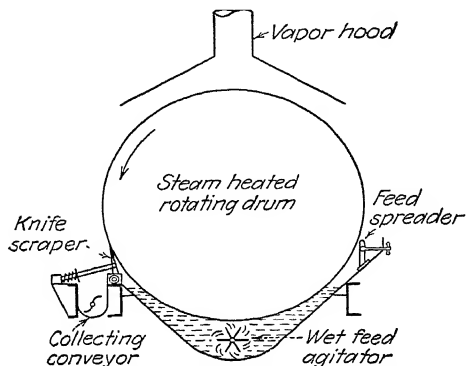


FIG. 8.—Drum drier. (Buffalo Foundry & Machine Co.)

atomization is produced by air injection of the material at high velocity, by discharging the solution or slurry through nozzles or centrifugal dispersion from a rotating disk or wheel. The dried material may be recovered from the bottom of the drying chamber or from collection equipment through which the spent drying gases are exhausted (Fig. 9).

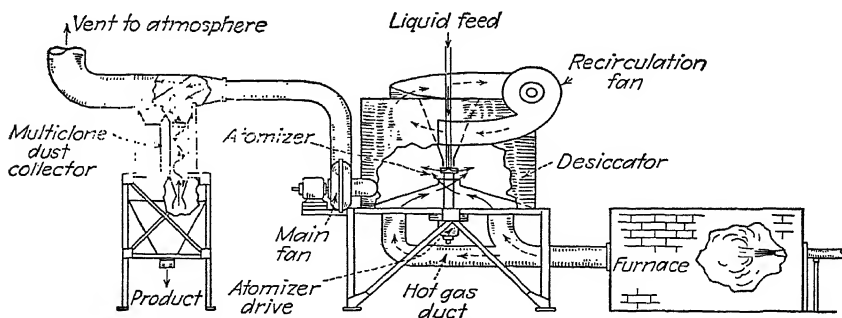


FIG. 9.—Turbulent-spray drier. (Western Precipitation Co.)

Batch Driers.—When the material to be dried is not continuously introduced to, and discharged from, the drying chamber, the operation is batch or intermittent drying. Such driers are known as compartment, tray, and tunnel driers.

Trays, pans, or cars are charged with undried material and put into a drying chamber which may be heated by any of the means used for continuous driers. When drying is complete, the dry charges are removed and fresh wet material is charged. When the drying chamber is so arranged that as a new charge is introduced a dry

charge is pushed from the drying chamber, operation may be considered continuous although wet feed is not introduced continuously at a regular rate.

Vacuum Driers.—The use of the vacuum drum-type drier has been mentioned. Other types such as spray, rotary, tray, and conveyer driers may be adapted for vacuum drying. Vacuum drying is usually confined to operations where comparatively low temperatures are necessary.

Drying Practice

Heat Expenditure in Drying.—The heat supplied for a drying operation is expended as (1) heat required to raise the temperature of the water, vaporize, and in some instances superheat the vapor; (2) heat to the material as discharged dry, above that when introduced undried; (3) heat to the exhaust air or gases above the ambient air temperature; (4) heat losses from the surfaces of the drying system to the surrounding atmosphere.

Theoretically, the effective heat utilization is that required to vaporize the water, and the remaining heat expenditure constitutes heat losses from the drying operation.

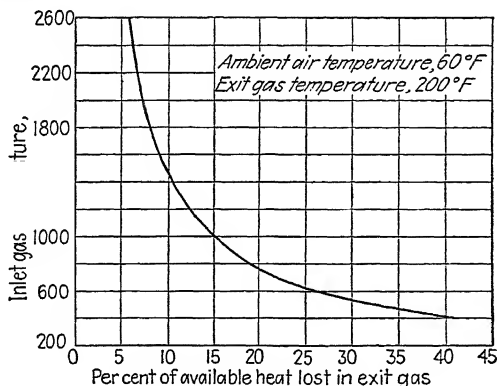


FIG. 10.—Drier heat loss in gases.

An exception is in instances where the sensible heat in the dried product is desirable or necessary for subsequent processing. In practice, the reduction of heat losses (heat not effectively utilized) is limited by (1) the type or classification of the drying system employed; (2) the physical characteristics of the material undergoing drying; (3) the initial and final moisture content of the material.

The term "drying efficiency" is in common use and expresses a ratio of effective heat utilization to total heat supplied. Frequently this expression is used as a means for comparing different types or classifications of driers with misleading effect. Comparisons of the effective heat utilization or thermal performance ratio for different methods or systems for drying should be made only for a particular drying requirement.

Whether the initial moisture content of the material is high or low, the temperature of the dried material is usually held within narrow limits. It is obvious that for low initial moisture a greater portion of the total heat supplied is lost as sensible heat in the dried material than when the initial moisture is high. This is true regardless of the type of drier used.

The exhaust humid air or gas from a drying system must be maintained above the temperature at which it would be saturated with water vapor. If the physical charac-

teristics of the material or the drying equipment limit the initial temperature, then the ratio of sensible heat loss to exhaust gas is greater than when high initial temperatures may be used (Fig. 10). For any drying operation using heated air or other gases as the heat source, the initial temperature should always be the highest possible without degradation or destruction of the material. Fuel cost is frequently the greatest single cost item in a drying operation, and the sensible heat in the exhaust gas can be a considerable portion of the total heat supplied.

At times, recycling a portion of the exhaust gas is practiced; this reduces heat loss and provides a control of humidity which may be desirable.

Classification of Materials.—Generally a material to be dried can be classified in one of three groups. The following tabulation is indicative of the type of driers in use with each group of materials. It will be seen that several types are adaptable to

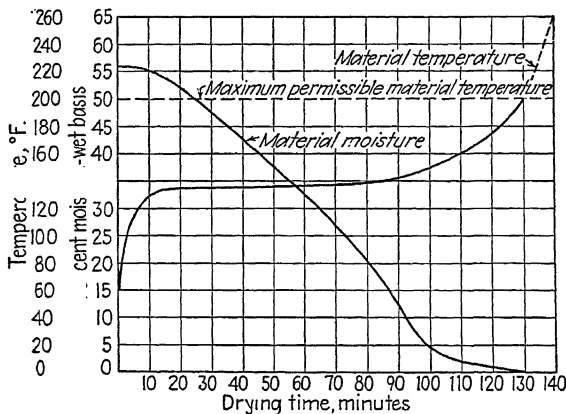


FIG. 11.—Drying chart—heat-sensitive material.

each material group, but it does not follow that each type performs equally well in every instance.

1. Materials insensitive to temperature and gases of combustion.
 - a. Direct-heat rotary driers.
 - b. Indirect-direct-heat rotary driers.
 - c. Direct-heat conveyer-type driers.
 - d. Dispersion-type driers.
 - e. Spray-type driers (high initial moisture and finely sized material).
2. Materials insensitive to temperature but sensitive to gases of combustion.
 - a. Indirect-heat rotary driers.
 - b. Indirect-heat conveyer-type driers.
 - c. Steam-tube rotary driers.
 - d. All direct-heat driers when preheated air is supplied.
 - e. Drum driers.
3. Materials sensitive to temperature.
 - a. Direct-heat rotary driers (warm air or tempered combustion gases).
 - b. Direct-heat conveyer driers (warm air or tempered combustion gases).
 - c. Steam-tube rotary driers (low-pressure steam).
 - d. Drum driers (low-pressure steam).

e. Vacuum driers (all types).

f. Rapid-vaporizing driers.

g. Batch driers.

Obviously there is some overlapping in the foregoing classification. Drum driers and spray driers are used where the undried feed is sufficiently high in moisture to ensure their proper functioning. Material particle sizing for any group limits the practicability of some types of driers.

Some materials in each group are more effectively dried by continuously returning a portion of dried or semidried material which is mixed with the wet feed to improve mechanical handling. This is generally done with the turbulent dispersion drier and frequently with the several types of rotary driers.

Drier Performance and Specifications.—The performance of drying systems varies widely with the limitations imposed by the materials to be dried. The prime limiting factors are water-vaporization requirements, temperature limitations, and drying time, all of which are related. Many other physical and chemical properties enter into drier performance. The amount of water to be removed varies from twenty times the weight of dry solids to as little as 1 per cent of the dry solids. Initial temperatures as high as 2600°F. are possible in some instances and as low as 100°F. in others. Drying time may be 30 sec. or several hours if diffusion is a limiting factor. Figure 11 illustrates the drying time of a heat-sensitive material in a constant-temperature indirect-heat rotary drier.

Table 1 presents general performance data for the various types of driers.

A proper analysis of each problem requires establishing in every instance drying specifications and in some instances mechanical specifications. Typical specifications include:

General information:

Capacity—dried product per hour or day (pounds or tons).

Moisture content—initial and final (specify wet or dry basis).¹

Specific heat.

Specific gravity or weight per cubic feet in bulk.

Particle size.

Combined and/or free moisture.

Temperature to which material can be heated without change.

Other physical and chemical limitations—plasticity or stickiness in wet state; discoloration or contamination from combustion gases; breakage or change of particle shape during drying; corrosive, abrasive, explosive.

Performance information:

Type of drier and size.

Fuel or steam consumption.

Temperatures of gas, air, or steam.

Velocity of gas or air in drying system.

Total power requirements.

Dust loss from system.

A recommended practice for testing drying equipment has been published by the American Society of Mechanical Engineers. It presents the procedure for establishing the performance of any drying-equipment installation.

¹ Wet basis is the percentage of water in the total weight of the wet material; Dry basis is the percentage of moisture referred to the weight of the moisture-free material.

$$\frac{\text{Per cent wet basis (weight of mixture)}}{\text{Weight of dry material}} = \text{per cent dry basis}$$

TABLE 1

Type of drier		Capacity factor	Heat consumption, B.t.u. per lb. H ₂ O evap.	Temperatures, °F.			Power, hp.	Remarks
				Heat source		Material		
Rotary	Direct heat	0.5-7.5 ^a	1600-3200	200-2600	100-350	110-600	2-200	Low capacity factor and high heat consumption with low inlet temperatures. High material temperature when desirable ^d
	Indirect heat	0.3-2.5 ^a	1300-4500	212-2000	120-350	150-300	2-80	Inlet temperatures 212 to 350°F. for steam-tube type. ^d Air and water-vapor temperature usually less than 200°F.
	Indirect-direct	2.0-4.0 ^a	1500-2400	1000-2000	130-200	150-300	2-125	Brick-encased type, inlet gas temperature under 1400°F ^d
Conveyor	Direct heat	1.0-10.0 ^a	2000-4000	200-1200	100-200	120-225	1-60	High capacity ratio for granular and lump materials and high inlet temperatures ^e
	Indirect heat	0.2-4.0 ^b	2000-5000	250-1500	100-200	120-225	1-60	Heat source may be infrared radiation
Drum	1.5-8.0 ^b	1300-2500	212-350	160-300	1-25	Capacity ratio high with high steam pressures. Film thickness controlled by rotating drum speed
Rapid vaporizing	Dispersion	75-4000 ^c	1600-2500	300-1300	140-200	130-180	2-80	Semidried material mixed and recirculated with wet feed improves capacity factor
	Spray	0.1-3.0 ^a	2000-5000	200-1200	120-200	130-180	5-100	Fine material and high temperatures for high capacity ratios. Gas velocities in drying chamber 50 to 200 ft. per min.
Batch	50-2000 ^c	1600-5000	200-1000	120-300	130-300	0-50	Frequently used when long drying time required because of low diffusion rate

^a Water evaporated per hour per cubic foot of drier volume.^b Water evaporated per hour per square foot of drier surface.^c Water evaporated per hour per drier unit.^d Exit gas or air velocities range from 50 to 900 ft. per min., depending upon particle size, specific gravity, and moisture.^e Gas or air velocity through or over drying bed 50 to 600 ft. per min., depending upon material and moisture.

For all direct-heat driers, capacity factor is affected by dust-collection requirements.

Drying Economics.—After determining the types of drying equipment that will deliver a satisfactory dried product, a proper selection becomes an economic consideration. This involves weighing for each type, as follows:


1. *a.* Initial installed cost including all auxiliary appurtenances necessary or desirable for operation of a particular type of drier.
b. Interest and amortization.
2. Operating costs.
a. Fuel cost.
b. Power cost.
c. Operating labor cost.
d. Maintenance cost (labor and material).
e. Dust-collecting cost.
3. Adaptability of the drier to provide satisfactory results with minimum interruptions to operation from any cause.

Drying costs vary as widely as drier performance, and drying costs can be established only following a specification and performance analysis.

CHAPTER III

CRUSHING AND GRINDING

BY HARLOWE HARDINGE¹

 **Fundamental Principles.**—In taking up the subject of crushing and grinding it is difficult to know just where to begin. The field is so broad and covers so many different applications that it is not an easy matter to cover the fundamental principles in a way that will apply to the majority of cases found in metallurgical practice. The object of crushing and grinding materials is twofold. In one case, the reduction is effected so that values may be extracted from the ore, and in the other case the entire mineral or product being ground is utilized for commercial purposes, such as feldspar and silica in the pottery trade, limestone and cement in the cement trade, coal for burning under boilers and in other furnaces, and other uses too numerous to mention here.

The dividing line between crushing and grinding is not sharp. Some machines may be considered to do crushing only, some grinding only, and others a combination of both. With but one or two exceptions, no one machine is capable of efficiently completing the entire crushing operation or the entire grinding operation if the range of reduction is large. In fact, to reduce a boulder from a size of 4 to 6 ft. in diameter down to a slime usually requires as many as five different crushing and grinding stages. Just how this process is conducted will be shown later in discussing the various types of machines used.

Ores and various materials to be crushed and ground vary in hardness, and it stands to reason that the soft ores are much easier to crush and grind than the hard ores. It also stands to reason that different types of equipment can be used to advantage in different cases. For soft ores, more complicated and less expensive machines may be employed, but for hard ores the crushers and grinding mills must be of very sturdy and simple construction in order to withstand the severe usage and strains encountered in effecting the reduction necessary. The ores encountered in metallurgical practice are, on the whole, considered hard as compared with many materials ground for commercial uses. This condition has somewhat simplified the selection of the type of crushing and grinding equipment required in metallurgical practice.

Many laws and formulas have been devised from time to time in an attempt to work out the whys and wherefores of crushing and grinding. There are today many who believe these laws fairly accurate and capable of solving most of the grinding problems. It is true that these laws will satisfy several conditions as found in practice, but, generally speaking, there are so many variables to consider that a formula covering one general condition would be so complicated that its use would be of little value when an attempt is made to apply it to another case. The study of laws is valuable from the standpoint of obtaining a better knowledge of what happens during the reduction process. In other words, their qualitative value is far greater than their quantitative value. The following two "laws" on crushing and grinding should be of assistance to the student who desires to study the theory from different angles, and

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of interest to all, as they show the conflict that exists when they are observed from a theoretical aspect.

Kick's and Rittinger's Laws.—Kick's law may be stated thus: The energy required for producing similar changes of form of similar bodies of the same substance varies as the volume or weight of these bodies.

Rittinger's law may be stated: The energy required to break down a given friable mass is proportional to the reciprocal of the diameter of the final product, assuming that the mass has all been reduced to one exact size, a condition that is only theoretically possible.¹

Kick's law, it will be noticed, deals with energy absorbed during deformation of the mass before rupture, while Rittinger's law bases the energy absorption upon actual fracture.

Figure 1 represents two pieces of ore of the same geometrical shape—in this case, cubes—placed between the surfaces of two crushing machines. The energy

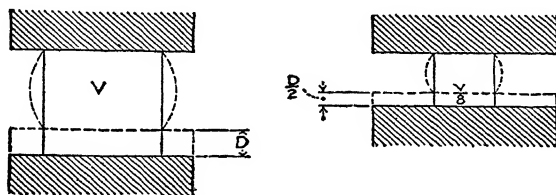


FIG. 1.—Kick's law.

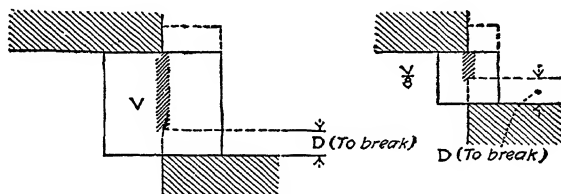


FIG. 2.—Rittinger's law.

that must be applied in each case to produce proportional deformation is the product of the average resisting force per square inch and the distance through which movement takes place. As shown in the figure, in this instance, the energy absorbed is proportional to the volume (Kick's law), and it can be similarly shown for the general case. On the gradual release of the extreme pressure, the energy absorbed is driven back through the machine producing the deformation and the body returns to its original shape. It should be noted that the body has been deformed only by a gradually increasing crushing force. The first increment of deformation does not require so much pressure as the last. If the body is deformed beyond its elastic limit, either the whole mass will be reduced to the molecular state by the freeing of their bonds from adjacent particles—a condition that could hardly occur—or the fracture takes place along the cleavage planes by the breaking down of some of the weaker bonds, and the new particles thus formed are free to resume their original shape insofar as they are not held between the machine surfaces. The energy given up by them is probably used in making fracture planes, which creates heat. This fact would indicate, then, that the energy absorbed according to Kick's law does not stay in the particle after pressure is released. It would be assumed, then, that the energy, according to

Kick's law, is absorbed by the creation of heat and not by crushing. This is correct according to the law of physics which states that all energy expended is finally dissipated as heat in some form.

Referring now to Fig. 2, assume that it is possible to hold similar cubes between two offset surfaces as shown and that forces are applied until the deformation shown by the dotted line is obtained. It will then be seen that only the molecules along the vertical center line are stressed and deformed. The mass of the cubes away from this surface would receive practically no pressure and, therefore, would not be deformed. The energy required in this case to produce rupture will be the product of the average resisting force per square inch of the area along which rupture takes place and the distance the two offset places move together (Rittinger's law). The average resistance to shearing varies as the deformation increases up to rupture. In order to reduce the mass to cubes, the amount of energy must be multiplied by 3, as three similar fracture planes must be made to produce cubes. It will not be hard to see that the distance through which the offset faces must move in either case is the same and is not proportional to the thickness of the piece. To break the molecular bond between adjacent particles would require the same movement regardless of the thickness of the piece. It will be noted that, if each of the half thickness cubes is sheared to produce cubes of half their size, the new surface presented will be double that formed when the single large cube is sheared to half-size cube, and also that the energy required for the smaller cube is double that required for the larger cube. This indicates that the energy applied to crushing is proportional to the surface produced, which would bear out Rittinger's law. Investigators now are of the opinion that Rittinger's law more nearly represents true conditions than does Kick's law.

Crushing.—Crushing as applied to metallurgy may be defined as the process of breaking down the ore or material by impact or direct pressure sufficient to cause fracture.

Crushing as employed in present-day practice is limited (1) to the size of the machine that can be used and (2) to the cost of reduction by this method. In large-scale operations, it is now possible to supply crushers that will handle rock as large as 8 ft. in diameter. Products economically disintegrated by the simple crushing action seldom are finer than $\frac{1}{2}$ in.—on rare occasions as fine as 10 or 12 mesh.

As the crushing action requires a great deal of force, the machines used must be very sturdy and be capable of withstanding severe shocks and strains. Of the many styles of crushers and rolls used in the past, as well as the literally hundreds of styles tried out, only a few types have survived.

The most efficient reduction consistent with economical operation and the proper size of units is about four diameters. In other words, when the maximum size piece is, say, 4 ft., most effective work can be accomplished if the discharge outlet is approximately 1 ft., although it is true that most crushers are designed to effect a greater reduction than this, since the loss in efficiency by increasing the reduction ratio is often offset by the fact that fewer stages will be required to bring the largest pieces to the finished size. In practice, this ratio runs anywhere from 4 to 8:1—seldom greater unless little regard is paid to economy.

Crushers are operated on the principle of a huge nutcracker. The rock to be crushed is dropped between two surfaces so inclined that the opening at the top is larger than the largest piece, and that at the bottom is adjusted to the size product desired. One of the surfaces is made to move a short distance toward the other, and in doing so, the rock being incapable of distortion, is broken under the tremendous pressure exerted. Since the distance the plate moves is comparatively slight, the total power required to do the crushing is far less than would at first be supposed. For example, to reduce run-of-mine rock from 6 ft. in diameter to 1 ft. in diameter, at

a rate of 400 tons per hr., about 250 hp. is required, but to reduce this same quantity from 1 ft. to 2 in. requires nearly 800 hp., and so on down the line.

Angle of Nip.—In nearly all types of devices used, the crushing surfaces form an angle. This construction allows the coarse pieces to be wedged and broken, and then still further reduced in the same machine, the amount depending upon the characteristics of the crusher.

This angle must be kept slight enough to prevent slippage of the rock wedged between the crushing surfaces. On the other hand, if the angle is too slight, a low ratio of reduction will result, and the device will not cover the commercial range of reduction required of it.

The angle of nip may be defined as that angle where the components of the forces, including friction, that act to hold the particles between the crushing surfaces are just balanced by the components of the forces that tend to snap the particles outward.

The safe, or proper, working angle of nip is naturally somewhat less than the critical. Theoretically, if there were no friction, the angle would have to be very slight to prevent slippage, as the only other force is that exerted by the weight of the piece in the wedge. This means, then, that the friction of the pieces against the crushing surfaces determines to a large extent the safe angle of nip. This angle, therefore, is subject to wide variation, depending on the material being crushed, the speed of approach, as well as the character of the crushing surfaces employed. In practice, however, it has been found that 31 deg. is about the safe angle for most conditions, and with most machines.

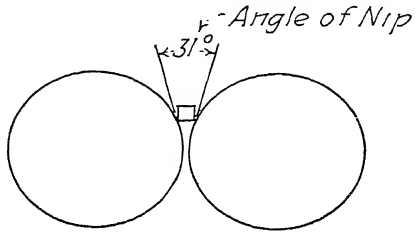


FIG. 3.—Angle of nip.

Where the crushing surfaces are round, such as rolls, which are described in detail later, the angle of nip varies at different sections. In this case, the angle of nip is that angle formed by the intersection of the two tangents, drawn between the two points of contact of the material wedged between both surfaces.

In either the jaw or the gyratory crusher, the angle of nip throughout the length of the crushing surfaces is nearly constant and the 31-deg. angle may be considered as the working angle, although a number of manufacturers have adopted a slighter one, in an endeavor to make it possible to crush many more kinds of material with one style of machine.

Rough rock, with sharp and abrasive characteristics, increases the frictional resistance, hence increases the effective angle of nip.

A corrugated or rough crushing surface tends to increase the angle of nip, since this surface increases the frictional resistance.

High speed of approach of the crushing surfaces decreases the effective angle of nip, since the rock fed to the surfaces is not immediately accelerated to the speed of the crushing surfaces and may fail to be caught.

The manner in which the material is fed also has a minor effect. Dry rock is usually better than damp rock, since, if damp, the clay or fine particles sticking to the rock tend to lower the coefficient of friction. On the other hand, if the rock is fed very wet the coefficient of friction is usually increased, owing to the washing away of the clay, which again exposes a clean, hard surface.

Types of Crushers.—The various types of crushers and crushing devices are known in practice as “primary breakers” and “reduction crushers.” The first class

includes the Blake type of jaw crusher and the gyratory crusher. These may be operated in series, where the range of reduction is large. In some cases, as many as three stages of so-called primary breakers may be used to advantage, the ultimate product being 1 to $\frac{1}{2}$ in.

The reduction crusher takes pieces as large as 4 to 6 in. in some cases, and in one operation may produce all passing $\frac{1}{2}$ in. Crushers of this type are the Dodge, reduction gyratory, swing hammer, and rolls. The last two types are also used to effect a reduction down to 10 or 12 mesh. While they are sometimes used to produce an even finer product, these cases are very unusual and are not to be generally recommended. Each type of crusher will be further discussed in detail, with its respective advantages and disadvantages.

Mill men and inventors have recognized the advantage of utilizing one or more of the mechanical principles found in the so-called crusher and adapting them to pulverize or at least encroach upon the field of a fine grinding device. A considerable amount of power would be saved and the character of product could then be more closely controlled than with the grinding devices now generally employed. To date, however, no such device has done more than fit into a highly limited application. The difficulty is that when attempts are made to affect the reduction by using the several

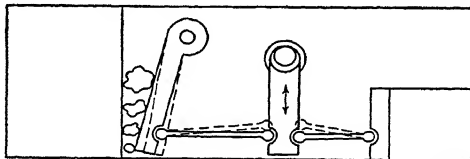


FIG. 4.—Double-toggle Blake crusher.

principles embodied in crushers, the crushing surface and motion must be greatly increased as the fineness is increased, usually in proportion to the new surface produced. The device then becomes excessively large at the point where the fine sizes are made and the movement, even though small at this point, becomes exceedingly hard to maintain mechanically. The surfaces are subject to wear and even a very minute change in clearance would have a major effect on the fineness produced. Any moisture present that tends to cause the fines to agglomerate or pack also causes serious complications, even more so than with reduction crushers now used close to their limiting size range when attempting to crush damp material.

Some of the grinding devices now used (see page 34) do utilize the basic crushing principles in one form or another, but, in doing so and to avoid the difficulties mentioned here, they sacrifice many of the advantages they have in the so-called crusher, and mechanically they are quite a bit different.

Jaw Crushers.—As stated above, jaw crushers are primarily used as coarse breakers. The double-toggle Blake type of jaw crusher is a coarse breaker and is now manufactured by many concerns in various modified forms. This crusher is built in many sizes. The largest takes boulders 8 ft. in diameter, and the smallest commercial size takes a feed around 3 or 4 in. The reduction ratio may be varied to suit the capacity and size requirements, within practical limits.

The principle of operation is simple. As can be seen from the diagram, there is one stationary plate and one movable plate. The movable plate is set at an angle to the stationary plate, with a large opening at the top to receive the coarse feed. The movable plate is also hinged at the top and oscillated by a toggle near the bottom. The incoming feed of coarse material is squeezed less near the top than the fine product near the bottom, but the movement is sufficient to cause fracture. The smaller

pieces resulting therefrom drop down and are further reduced in the narrowing opening until they are finally crushed sufficiently to drop out at the opening between the plates at the bottom.

The movable jaw is strongly hinged at the top. The pitman, which is the central member, is made to move up and down by means of an eccentric on the drive shaft. At the other end of the pitman are toggles so adjusted that when the pitman moves up and down the toggles are made to exert a lateral movement which is transmitted to the swinging plate. On the drive shaft is a large flywheel to assist in the operation during the crushing stroke. The movement of the jaw exerts a tremendous crushing force, and therefore the toggles are made the weakest member in the crusher, for if tramp iron or some hard nonfriable material accidentally gets between the jaws, the toggles will break and cause no further damage. Extra toggles are kept on hand, and it is a simple matter to replace them.

A spring is used to keep the movable jaw against the toggles at all times, as otherwise the rapid oscillating movement would be enough to jar the toggles loose and they would then drop out.

All wearing parts are made replaceable. The jaw plates are of alloy steel, usually manganese steel, so as to withstand the severe wear to which they are subjected.

The size of the discharge opening, as well as the amount of swing, may be adjusted. Adjustment of the opening in the type of crusher illustrated is effected by varying the thickness of shims between the frame and the tail block. The length of stroke is varied by adjustment of this same tail block up or down by means of other shims. Raising the tail block increases the angle of the rear toggle, which, in turn, increases the stroke. In the same way, lowering the block by adding shims decreases the stroke.

TABLE 1.—BLAKE-TYPE DOUBLE-TOGGLE JAW-CRUSHER DATA

Jaw opening, in.	Tons per hr. to sizes given below (approximate)						Approximate weight, lb.	Size of pulleys, in., single drive	R.p.m.	Hp. (approx.)
	In.	Tons per hr.	In.	Tons per hr.	In.	Tons per hr.				
66 × 84	10	479	12	572	14	682	550,000	180 × 36	85- 90	250-300
56 × 72	10	495	12	594	14	693	350,000	144 × 36	115-125	175-225
48 × 60	7	286	8	330	10	413	240,000	120 × 24	125-135	140-175
42 × 54	6	231	8	319	10	396	195,000	90 × 36	140-150	125-150
40 × 42	7	187	9	242	12	308	123,000	84 × 24	160-175	110-140
36 × 48	6	209	8	286	10	358	123,000	84 × 20	160-175	115-140
30 × 42	4	127	5	160	6	193	116,000	84 × 18	160-175	90-115
30 × 42	5	143	6	176	7	220	72,000	70 × 24	185-200	85-100
24 × 36	3	77	4	99	5	138	68,000	70 × 18	210-225	75- 85

It is found advisable in practice to adjust the swing of the jaw to suit both the nature of the material and the degree of fineness required. For coarse crushing and maximum capacity, a long stroke is desirable. For fine crushing, it is best to use a short stroke in order to prevent the finer pieces from packing between the jaws.

Advantages.—The outstanding advantage of the jaw crusher over the gyratory, or any other type of crusher, is that it has a larger feed opening, other conditions being equal, which means that coarser material may be fed to this crusher. The head room required in a jaw crusher is lower than that of the equivalent gyratory. The wearing plates are quickly and easily removed. The moving parts are more accessible than

includes the Blake type of jaw crusher and the gyratory crusher. These may be operated in series, where the range of reduction is large. In some cases, as many as three stages of so-called primary breakers may be used to advantage, the ultimate product being 1 to $\frac{1}{2}$ in.

The reduction crusher takes pieces as large as 4 to 6 in. in some cases, and in one operation may produce all passing $\frac{1}{2}$ in. Crushers of this type are the Dodge, reduction gyratory, swing hammer, and rolls. The last two types are also used to effect a reduction down to 10 or 12 mesh. While they are sometimes used to produce an even finer product, these cases are very unusual and are not to be generally recommended. Each type of crusher will be further discussed in detail, with its respective advantages and disadvantages.

Mill men and inventors have recognized the advantage of utilizing one or more of the mechanical principles found in the so-called crusher and adapting them to pulverize or at least encroach upon the field of a fine grinding device. A considerable amount of power would be saved and the character of product could then be more closely controlled than with the grinding devices now generally employed. To date, however, no such device has done more than fit into a highly limited application. The difficulty is that when attempts are made to affect the reduction by using the several

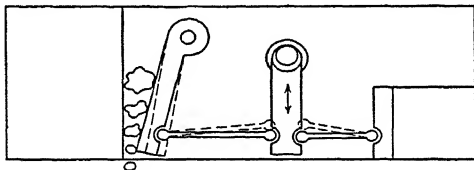


FIG. 4.—Double-toggle Blake crusher.

principles embodied in crushers, the crushing surface and motion must be greatly increased as the fineness is increased, usually in proportion to the new surface produced. The device then becomes excessively large at the point where the fine sizes are made and the movement, even though small at this point, becomes exceedingly hard to maintain mechanically. The surfaces are subject to wear and even a very minute change in clearance would have a major effect on the fineness produced. Any moisture present that tends to cause the fines to agglomerate or pack also causes serious complications, even more so than with reduction crushers now used close to their limiting size range when attempting to crush damp material.

Some of the grinding devices now used (see page 34) do utilize the basic crushing principles in one form or another, but, in doing so and to avoid the difficulties mentioned here, they sacrifice many of the advantages they have in the so-called crusher, and mechanically they are quite a bit different.

Jaw Crushers.—As stated above, jaw crushers are primarily used as coarse breakers. The double-toggle Blake type of jaw crusher is a coarse breaker and is now manufactured by many concerns in various modified forms. This crusher is built in many sizes. The largest takes boulders 8 ft. in diameter, and the smallest commercial size takes a feed around 3 or 4 in. The reduction ratio may be varied to suit the capacity and size requirements, within practical limits.

The principle of operation is simple. As can be seen from the diagram, there is one stationary plate and one movable plate. The movable plate is set at an angle to the stationary plate, with a large opening at the top to receive the coarse feed. The movable plate is also hinged at the top and oscillated by a toggle near the bottom. The incoming feed of coarse material is squeezed less near the top than the fine product near the bottom, but the movement is sufficient to cause fracture. The smaller

pieces resulting therefrom drop down and are further reduced in the narrowing opening until they are finally crushed sufficiently to drop out at the opening between the plates at the bottom.

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A spring is used to keep the movable jaw against the toggles at all times, as otherwise the rapid oscillating movement would be enough to jar the toggles loose and they would then drop out.

All wearing parts are made replaceable. The jaw plates are of alloy steel, usually manganese steel, so as to withstand the severe wear to which they are subjected.

The size of the discharge opening, as well as the amount of swing, may be adjusted. Adjustment of the opening in the type of crusher illustrated is effected by varying the thickness of shims between the frame and the tail block. The length of stroke is varied by adjustment of this same tail block up or down by means of other shims. Raising the tail block increases the angle of the rear toggle, which, in turn, increases the stroke. In the same way, lowering the block by adding shims decreases the stroke.

TABLE 1.—BLAKE-TYPE DOUBLE-TOGGLE JAW-CRUSHER DATA

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	In.	Tons per hr.	In.	Tons per hr.	In.	Tons per hr.				
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56 × 72	10	495	12	594	14	693	350,000	144 × 36	115-125	175-225
48 × 60	7	286	8	330	10	413	240,000	120 × 24	125-135	140-175
42 × 54	6	231	8	319	10	396	195,000	90 × 36	140-150	125-150
40 × 42	7	187	9	242	12	308	123,000	84 × 24	160-175	110-140
36 × 48	6	209	8	286	10	358	123,000	84 × 20	160-175	115-140
30 × 42	4	127	5	160	6	193	116,000	84 × 18	160-175	90-115
30 × 42	5	143	6	176	7	220	72,000	70 × 24	185-200	85-100
24 × 36	3	77	4	99	5	138	68,000	70 × 18	210-225	75-85

It is found advisable in practice to adjust the swing of the jaw to suit both the nature of the material and the degree of fineness required. For coarse crushing and maximum capacity, a long stroke is desirable. For fine crushing, it is best to use a short stroke in order to prevent the finer pieces from packing between the jaws.

Advantages.—The outstanding advantage of the jaw crusher over the gyratory, or any other type of crusher, is that it has a larger feed opening, other conditions being equal, which means that coarser material may be fed to this crusher. The head room required in a jaw crusher is lower than that of the equivalent gyratory. The wearing plates are quickly and easily removed. The moving parts are more accessible than

the gyratory. Also variation in the length of stroke consistent with the size and character of product can be readily obtained. First cost compared with gyratory crushers of equivalent size is usually less. It can withstand very hard, tough materials and can handle moist and sticky materials.

Disadvantages.—The power required to operate a jaw crusher is somewhat greater than that required by the equivalent gyratory. The jaw crusher has a greater tendency to choke under heavy feed than the gyratory, partly because of the large

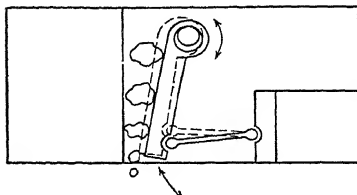


FIG. 5.—Single-toggle crusher.

opening as compared with the capacity, and partly because of the shape and motion. It will not crush to finer sizes economically.

Single-toggle Jaw Crushers.—The single-toggle jaw crusher (Fig. 5) causes the jaw to be moved at the top as well as at the bottom. The eccentric is mounted in the top of the swinging jaw. One end of the toggle is pivoted in a block on the stationary back frame and above the other end of the toggle placed in a seat in the back of the moving jaw. Rotation of the eccentric shaft moves the jaw in and out, and up and down. Shims or wedge blocks behind the toggle block on the frame are employed

TABLE 2.—SINGLE-TOGGLE JAW-CRUSHER DATA¹

Size of crusher (feed opening), in. . .	9 × 16	9 × 30	13 × 24	18 × 30	24 × 36
Net weight of crusher, lb.	5,400	10,100	9,700	20,250	40,000
Horsepower required.	8 to 15	15 to 25	20-30	50-60	75-100
Drive pulley:					
Diameter, in.	33	38	38	48	60
r.p.m.	420	350	350	275	250
Capacity, tons per hr.					
With ½-in. discharge opening. . . .	3-4				
With ¾-in. discharge opening. . . .	5-6	12-15			
With 1-in. discharge opening. . . .	6-8	15-18			
With 1½-in. discharge opening. . . .	10-12	23-28	20-25		
With 2½-in. discharge opening.	25-30	30-40	
With 3-in. discharge opening.	28-35	35-45	
With 4-in. discharge opening.	70- 80
With 6-in. discharge opening.	90-100

¹ From Smith Engineering Works *Bulletin*.

NOTE: The capacities are in tons of 2,000 lb. and are based on limestone weighing loose 2,600 lb. per cu. yd. The horsepower varies with the size of the product, capacity, and hardness of rock. For close settings, all undersize material should be removed from the feed so as to eliminate packing and excessive wear on the jaw dies. No crusher, when set to any given discharge opening, will produce a product all of which will pass a screen opening of the same dimensions as the given discharge opening. The amount of oversize will vary with the character of rock.

to regulate the jaw opening. Lowering the stationary toggle block shortens the in-and-out movement of the jaw at the bottom which is advantageous when a fine product, at less capacity, is desired.

Operating data of Telsmith-Wheeling jaw crushers, which is one of many designs of this type, are given in Table 2.

Advantages.—It produces a more uniform product than the double-toggle type. The construction is simple, and for the floor space required, the capacity is high. It is well suited to moderate sized operations.

Disadvantages.—It has not proved practical in large sizes because of heavy weight of swing jaw. The eccentric bearings must take all the weight of the jaw and direct pressure caused by the crushing action.

Dodge Crusher.—The Dodge crusher (Fig. 6) is another type of jaw crusher, but it delivers a more uniform product than the Blake type.

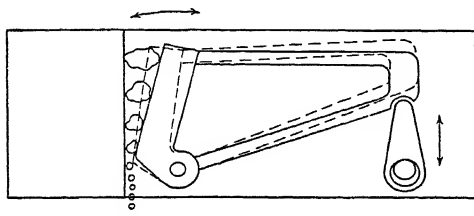


Fig. 6.—Dodge crusher.

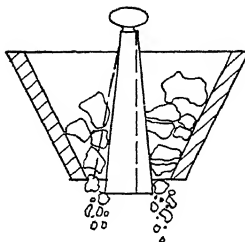


Fig. 7.—Gyratory crusher.

The principle of operation is illustrated in the diagram. One plate is stationary, the other is hinged at the bottom, causing greater motion at the feed opening than at the discharge. This construction prevents large pieces from dropping through on the backstroke, as occurs with the Blake type of jaw crusher.

Both jaw surfaces are covered with wearing plates that are removable. The movable jaw is hinged by a large pin at the bottom capable of withstanding heavy strain. This section is moved by a strong lever, actuated by an eccentric on the main drive shaft. A heavy flywheel is used to balance the operation and to reduce the power consumption. Adjustment is made by moving the packing blocks of the jaw shaft backward or forward by means of shims. Table 3 gives data on Allis-Chalmers Dodge crushers.

TABLE 3.—DODGE CRUSHER DATA

Size or jaw opening, in.	Capacity, tons (2000 lb.) per hr. (1 ton = 20 cu. ft.) capacity based on ring size				Approx. hp. required	Size of pulleys, in.	R.p.m.	Weight complete, lb.
	Size of product, in.							
	½	¾	1	1½				
4 × 6	¼	½	1	..	3	16 × 4½	275	1,100
7 × 9 1	2	3	3	6	24 × 5½	235	3,250
8 × 12 1½	3	4	4	10	30 × 6½	220	5,400
11 × 15 2	4	6	6	15	32 × 10	200	13,500

Advantages.—The Dodge type of reduction crusher is the simplest and least expensive of the standard types. It makes a uniform product without requiring constant adjustment.

Disadvantages.—It will choke if fed at too rapid a rate, and it does not have the capacity, for either the floor space taken or the power consumed, that several other types have.

Gyratory Crushers.—The gyratory crusher, like the jaw crusher, is used as a primary breaker. This crusher is also constructed in many sizes to suit conditions, and for large-scale operations, particularly where the run-of-mine work is large. These crushers may be run in series, and the second and possibly the third stage may still be considered as primary crushing if the reduction does not go below $1\frac{1}{2}$ in. maximum size piece.

The principle of operation of the gyratory crusher is better understood by a study of the schematic diagram (Fig. 7). The outer section is a truncated cone open at the top and bottom. A heavy shaft hinged at the top is suspended in the center of the cone; the bottom is gyrated, and this causes the surface of the central shaft to approach the cone at different points during each cycle. The rock to be crushed can be dumped on top of the crusher and may lodge in any portion of the annular opening and be crushed by the same pinching action as described for the jaw crusher. It should be noted, however, that every movement of the gyratory crusher is capable of crushing.

The central shaft is rigidly hinged at the top on a swivel joint. Fastened to the shaft is the crushing head, made so that it can be readily replaced when worn, as is the case with the shell liners which are located on the stationary member. At the bottom of the central shaft is located the driving mechanism, which consists of an eccentric, on which is located a gear. This gear is driven by a pinion fastened to the pulley shaft. When the eccentric rotates, it causes the bottom of the shaft to describe a small circle. The oscillating movement of this shaft near the top changes the distance once each revolution between the liner on the shaft and the adjacent point on the shell liners. The leverage thus secured by this principle is sufficient to produce a great crushing force and reduce any hard, friable rock. The product dropping through the annular space at the bottom is caught in a chute and discharged at the side of the crusher.

In the event that tramp iron or any other substance impossible to fracture drops into the crusher and is too large to pass through, a shear pin is used, which is the weakest member and will break first, and so avoid breaking the shell or other vital part. The breaking pin is located on the pulley and transmits the power to a keyed collar on the shaft. If the pin breaks, the pulley will then revolve on the shaft as an idler, and no power will be transmitted to the crusher.

Adjustment of the size of opening is made by raising or lowering the central shaft by means of the adjusting nut located on the thread on the top of the shaft. It will be noted that the wearing head on the shaft extends below the shell liners. Consequently, raising the central shaft decreases the opening, with a consequent decrease in the size of product produced, and vice versa, for a coarse product, the shaft is lowered.

Advantages.—The gyratory crusher requires less power than the equivalent jaw crusher, as all motion is available for crushing, whereas in the jaw crusher there is no crushing action on the backstroke. The gyratory crusher has a large capacity in proportion to the floor space occupied. It is capable of standing a choke feed to better advantage than the jaw crusher. There is less vibration in operating. Slabby material is effectively crushed.

Disadvantages.—The head room of a gyratory crusher is considerably more than the equivalent jaw crusher. The first cost is usually higher when all factors are

considered. The gyratory crusher will not take so large a feed as the corresponding jaw crusher. Wet sticky material tends to clog in this type of crusher. Table 4 covers data on Traylor primary gyratory crushers.

TABLE 4.—DATA FOR GYRATORY CRUSHERS AS PRIMARY CRUSHERS

Size of each feed opening, in.	Fine setting		Coarse setting		Size of driving pulley, in.	R.p.m.	Hp. required	Approx. wt. of crusher, lb.
	Small-est size of product, in.	Capacity, tons (2000 lb.) per hr.	Largest size of product, in.	Capacity, tons (2000 lb.) per hr.				
2½ × 14	¾	¾	¾	¾	8 × 3	500	2-4	550
8 × 37	1	16	2¼	36	24 × 10	450	12-25	15,500
12 × 46	1½	28	3½	75	36 × 12	365	25-45	28,000
14 × 55	2	52	4½	115	40 × 14	345	40-75	38,000
16 × 63	2¼	93	5	185	48 × 18	340	60-100	53,500
20 × 80	3	157	5½	330	48 × 20	330	90-150	93,000
26 × 100	3¾	289	6½	525	52 × 24	320	125-200	153,000
30 × 118	4	350	7	700	52 × 24	310	125-200	168,000
36 × 136	4½	595	8	1056	60 × 28	300	150-250	300,000
42 × 153	5	930	8½	1581	66 × 32	275	175-275	360,000
48 × 166	5½	1158	9	1890	72 × 36	250	200-350	500,000
60 × 210	7	1678	10	2400	80 × 40	200	250-450	780,000
72 × 242	8	2572	12	3432	92 × 48	175	300-500	1,250,000

Reduction Gyratory Crushers.—Reduction gyratory crushers have met wide use in the intermediate- and fine-crushing range, because of their high capacity at reasonably large ratios of reduction. Where materials are friable and not sticky, this type of crusher is being generally adopted for crushing down to a size where it begins to compete with rod or ball mills, which is usually at about ¾ to ½ in., or somewhat finer if the material undergoing reduction is not to be subsequently ground. Crushers in this category are basically high speed and closely set as compared with the standard gyratory crusher.

Symons Cone Crusher.—The Symons cone crusher (Fig. 8) is a gyratory reduction crusher that employs a modified principle of operation different from the standard form of gyratory reduction crusher. The head gyrates much the same as the standard gyratory crusher but travels several times as far and at much greater speed. The pieces being crushed pass through the cavity in a series of bounces because of the high speed of the head, which, as it recedes from the closed position and from under the material, permits the material to fall vertically a short distance before the cone again comes up for another crushing impact. Because of the short

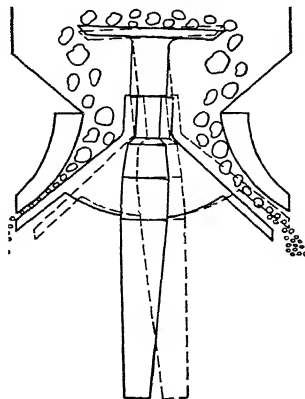


FIG. 8.—Symons cone crusher.

time cycle, the material is subjected to a number of crushing blows before it can pass through and must pass a zone where the crushing surfaces are parallel for a sufficient distance to ensure at least one crushing impact before the material is discharged. All pieces will at least have one dimension no larger than the minimum opening in the parallel zone. This zone, or closed side, thus regulates the size of product and not the open side. Table 5 gives capacity, powers, and sizes of Symons cone crushers.

Advantages.—The Symons cone crusher has the same advantages as gyratory crushers. It has a large ratio of reduction, is economical in power consumption, and delivers a uniform size of product at high capacity per floor space and power consumption. Maintenance is low.

Disadvantages.—It is high in first cost, particularly for relatively low capacities where the Dodge type of crusher can be employed. It tends to choke with damp or clayey materials, and it must be regularly adjusted to maintain a specified size.

TABLE 5.—SYMONS CONE CRUSHER DATA

Size of crusher	Capacities, tons (2000 lb.) per hr. at indicated discharge setting A with material weighing 100 lb. per cu. ft.								Full load, r.p.m.	Hp.	Shipping weight, lb.
	$\frac{3}{16}$ "	$\frac{3}{8}$ "	$\frac{1}{2}$ "	$\frac{3}{4}$ "	1"	$1\frac{1}{2}$ "	2"	$2\frac{1}{2}$ "			
20"	8	15 15	20 20 30 40	650	20 - 25	7,700
2'	20 20	25 25	35 35 45 60	575	25- 30	10,000
3'	35	40 40	70 70 80 90 95	580	50- 60	21,000
4'	60	80	120 120	150 150 177 185	485	75-100	35,000
4 $\frac{1}{4}$ '	100	140 140 140 160 160 185 190	485	125-150	45,000
5 $\frac{1}{2}$ '	200	275 275 275 340 340 375 375 450 450	485	150-200	83,000
7'	330	450 450	600 600 600 800 800 900	435	250-300	140,000

Hammer Mills.—The hammer mill (Fig. 9) is a distinct type of reduction crusher and is adaptable to conditions where the rock to be crushed is friable, but not abrasive. Limestone is an example of the class of rock that is readily crushed. The ratio of reduction in a hammer mill can be increased over that of the other crushers mentioned.

The principle of operation of the hammer mill is best understood by a study of the diagram. Short bars are hinged to a rapidly revolving shaft inside a housing. The

rock is fed into a hopper on the down side of the revolution. As the tips of the hammers move much faster than the pieces of rock fall, they strike the pieces a blow that sends the shattered rock to the casing with sufficient force to cause still further disintegration. Those pieces that are fine enough drop through the slots. The others are carried around the casing, once more dashed against the wearing plate or the incoming rock itself, and broken down to a size that drops through the holes.

There is a removable wearing plate below the feed hopper which takes a considerable portion of the impact of the rock as it is struck down by the hammers. Also the bottom is slotted through a comparatively thick section in order to increase the life of this section and prevent frequent renewals. The hammers are hinged, so that when an extra large piece drops into the crusher and is not broken down the first time the hammer will ride over it and not cause breakage of the casing or of the hammers themselves. Fineness may be adjusted by varying the size slot through which the product discharges.

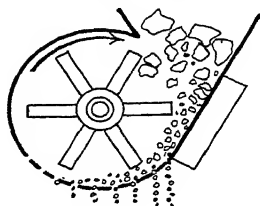


FIG. 9.—Hammer mill.

TABLE 6.—HAMMER MILL DATA¹
Sizes and Capacities of Swing-hammer Mills

Size of machine, in.	Approx. hp.		Approx. weight, lb.	Feed opening, in.		Speed, r.p.m.	
20 × 12	12- 20		3,000	11 × 11		1500-1800	
24 × 20	30- 40		4,800	13 × 20		1400-1600	
36 × 24	60- 75		9,600	18 × 24		900-1200	
42 × 36	100-125		17,200	21 × 36		700-1000	
42 × 66	175-250		36,000	22 × 66		700-1000	

Capacities, Tons per Hour						
Size of machine, in.	Bar opening, in.					
	Limestone		Coal		Gypsum	
	$\frac{1}{8}$	1	$\frac{1}{8}$	1	$\frac{1}{8}$	1
20 × 12	2- 2½	4- 6	2- 3	6- 8	3- 3½	5- 7
24 × 20	4- 5	10- 15	6- 8	18- 20	6- 7	15- 20
36 × 24	10-12	25- 35	14-16	45- 60	15-18	30- 35
42 × 36	20-25	60- 70	30-40	85-100	30-35	60- 70
42 × 66	50-60	100-175	75-85	175-220	65-75	125-140

¹ From Jeffrey Mfg. Co. Catalogue.

When the hammers wear, a number of means have been provided to counteract their tendency to become short. In one type, holes are drilled in the hammer at short distances apart. When the hammer becomes short, the pin is placed through a hole nearer the opposite end of the bar. Another method provides an adjustable tapered tip on the end of the hammer that is slid out as the hammer wears.

Advantages.—The swing-hammer crusher is capable of effecting a larger ratio of reduction than other types of reduction crushers. It has a large capacity for the floor space and has a smaller first cost than other reduction crushers.

Disadvantages.—The speed at which the hammers rotate is far greater than any other type of crusher. This increases the wear on moving and wearing parts and is dangerous where much tramp iron is encountered. Except on soft, nonabrasive materials, the wear on hammers and casing is excessive. This crusher is unable to reduce moist or sticky materials satisfactorily.

Rolls.—Rolls may be classified as reduction crushers. They differ in principle from any of the types already described and are primarily adapted for use where finer crushing is desired than the regular reduction crusher can accomplish.

The most general use of the roll is for feed around 3 to $\frac{1}{2}$ in. Rolls are built to handle feed as coarse as 8 and 9 in., reducing this size to approximately 2 in. Others take feed as fine as $\frac{1}{4}$ in. and deliver a product as fine as 10 mesh, but this operation approaches that of grinding. Rolls are advocated for fine reduction only in exceptional cases now, as various types of grinding mills can accomplish the work to better advantage.

Rolls, like crushers, are sometimes operated in series, since the ratio of reduction for rolls should not exceed 4:1. As with other types of crushers, coarse reduction can be accomplished with less floor space and power consumption than the same tonnage of finer reduction.

The principle of operation of rolls is that of two cylinders with faces and axes parallel revolving so that the adjacent faces rotate downward and the rock dropped between them is broken by the pinching action thus produced. This action is illustrated in the diagram to the left. Either or both rolls may be positively driven. The capacity varies with the size of feed, product desired, peripheral speed of shell, and width of face, but only slightly with the hardness of the rock if operated under proper conditions.

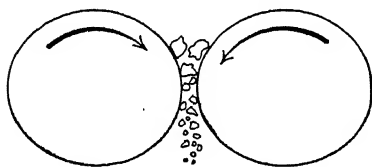


FIG. 10.—Roll crushing.

Figure 10 shows a diagrammatic view of a set of rolls. In practice, the bearing bases of one roll are made solid with the frame and the other set is on a sliding base. The roll on

the sliding base is held in position by heavy springs pulling tension rods fastened to the movable bearings. The rolls are kept apart by shims placed between the bases.

To provide for wear, the shell of the rolls is made replaceable. In one commonly used type, the shell is removed by unbolting a wedged-shaped annular ring extending part way under the shell and what is known as the fixed heart, which is the inner portion of the roll. Guides over the rods, where the material is introduced, are so arranged that the feed is directed into the space between the rolls. A housing completely covers the roll cylinders in order to eliminate dust as well as keep pieces that slip out or snap back from covering the immediate surroundings. Bearings are made very heavy in order to withstand the heavy pressures that are exerted when the rolls are fed to capacity. The springs holding one roll near the other are of sufficient strength that they are not further compressed during ordinary operation, unless tramp iron or other very hard substance drops between the faces. In that case, the rolls will spread apart, allowing the tramp iron to drop through. The springs will then pull the roll back into its former operating position without interrupting the functioning of the rolls.

As rolls have a tendency to corrugate, thus making it impossible either to secure a uniform product or a maximum capacity, various means have been employed in an

attempt to correct this fault—a fault, by the way, which is the main reason that rolls cannot be used to advantage for medium-fine grinding. The method in most general use is to make one roll adjustable laterally, so that when corrugation starts this roll is moved over slightly. Thus different surfaces are placed opposite one another, and the tendency to corrugate is somewhat reduced. One type of roll provides for this change to take place continuously by means of reduction gears operating a thrust bearing. This action is called “fleeting.” Another method, adopted in the field, is to press a small flat grindstone against the ridges in the rolls, thus wearing them down. This method requires a good deal of attention, but is sometimes the only expedient other than to stop the rolls entirely and either turn down the faces in the machine shop, or replace them with new rings or shells.

Advantages.—Rolls with smooth surfaces give the most uniform grade of product that it is possible to obtain from any type of crushing device, are economical in power consumption as compared with the work performed, and require only a small amount of head room or floor space for the capacity. They fill a gap between reduction crushing and grinding better than any other device.

Disadvantages.—Rolls require constant attention to keep them efficient. Corrugations often occur which greatly decrease the capacity and uniformity of product. The repair cost on rolls is higher than on other crushing devices used to accomplish the same work. While rolls can be operated to deliver a product finer than $\frac{1}{4}$ mesh, it is not usually found economical to do so.

Roll Data.—The theoretical capacity is obtained by computing the cubical content of the ribbon of material passing through the roll. The data required are the peripheral speed of the roll, width of face, distance apart of the two cylinders, and weight per cubic foot of the material passing through. In practice, because of the unequal character of the feed, not more than one-quarter of the theoretical capacity should be taken as the actual capacity. The following is the formula generally used:

$$\frac{D \times W \times S}{1728 \times 4} = \text{effective capacity in cubic feet per minute}$$

where D = distance roll faces are separated in inches.

W = width of face in inches.

S = peripheral speed in inches per minute.

Roll Crushers.—A modification of the standard roll is the so-called “roll crusher,” for crushing relatively soft materials like bituminous coal, salt, and alum. The single-roll crusher consists of a roll equipped with heavy teeth or deep sharp corrugations to increase the angle of nip or actually dig into the coarse lumps as they enter the opening above the roll. Spaced from the roll is a curved plate with the bottom bending in toward the bottom of the roll and practically touching it. In operation, the feed is dragged downward by the toothed or spiked roll. The material is crushed as it is wedged into the decreasing space as the bottom of the roll is reached, where it is discharged at a size not greater than the gap between the flat portion of the roll and face of the stationary plate. The capacity is high and the ratio of reduction large. The roll cannot produce a fine product owing to the spacing between adjacent teeth, actually not finer than 1 to $1\frac{1}{2}$ in., but feeds as coarse as 24-in. lumps can be fed and reduced to this size in one unit.

Double-roll crushers are similar to the standard double roll except that the rolls are equipped with teeth, spikes, or corrugations to increase the angle of nip and ratio of reduction, but in any case only materials relatively easy to crush can be successfully handled in these types with reasonable maintenance.

Figure 11 is a chart covering the more popular sizes of rolls showing relationship of feed, product, speed, and roll diameter. Table 7 gives data on Traylor rolls.

TABLE 7.—CRUSHING ROLL DATA¹

Size of roll, diameter by face, in.	Approx. capacity, tons (2000 lb.) per hr. ²			Rolls without automatic lateral-adjustment mechanism		R.p.m. (usual range)	Hp. required
	Size prod., in.	Max. feed, in.	Capacity, tons	Movable and fixed pulleys, in.	Approximate shell wt., lb.		
9 × 10	1/8	1/2	2	18 × 6	2,400	300-350	6
18 × 10	1/8	5/8	5	24 × 8	5,600	250-300	8
24 × 12	1/8	7/8	10	48 × 6	11,800	180-230	10
30 × 16	1/8	1	20	60 × 10	20,000	130-180	15
36 × 16	1/4	1 1/4	35	72 × 10	59,300	100-150	40
42 × 18	1/4	1 1/2	45	72 × 12	65,700	95-120	55
48 × 20	3/8	1 7/8	60	84 × 12	81,500	75-105	65
54 × 24	3/8	2	75	84 × 16	103,200	70-95	75
60 × 30	1/2	2 3/8	115	96 × 16	154,000	65-85	100
72 × 36	1/2	3	155	108 × 18	248,800	50-75	150

¹ From Traylor Engineering & Mfg. Co. *Bulletin*.² Capacities are based on materials weighing 100 lb. per cu. ft. when crushed.

Notes on Crushing.—In selecting a crushing device for a particular use, a number of factors should be considered to ensure that the right type and size will be chosen. The comments made here deal primarily with the treatment of friable materials.

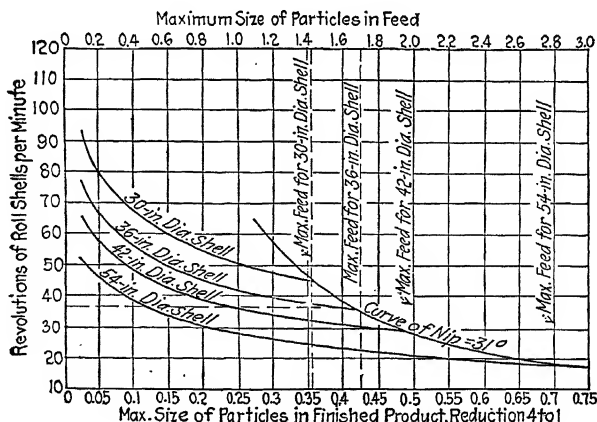


FIG. 11.—Size of feed for rolls.

Feed size is the first consideration. Where the capacity is large and the feed sizes also are large, a double-toggle Blake type of crusher is usually indicated. The feed opening of a Blake-type crusher is larger than the equivalent size gyratory. Less

headroom is required, which is a consideration where very large units are called for. The capital cost is also less as a rule.

Jaw crushers are also indicated where the tonnage required is moderate, say 100 tons per shift, and the size range of the feed is appreciable, say from 30 in. to dust. Good practice is to select a crusher of sufficient opening to take about 90 per cent of the total feed. That too large to pass the opening can be sledged by hand or mud capped until it breaks small enough to enter the crusher. For larger tonnages, between 300 to 500 tons, a jaw crusher to handle all the feed would be more economical than to increase the amount of labor needed to handle sizes too coarse to be fed directly. For still larger capacities on the order of 2500 tons, a gyratory of large opening might be best since a jaw crusher of equal capacity at a given setting might have much too large an opening. Yet again for still larger tonnage, a large jaw followed by a gyratory would be the most economical and satisfactory arrangement. Where the product is ultimately to be ground, then a third stage using a reduction gyratory or rolls is usually indicated to bring the feed to the grinding mills down to the economical size, depending on the material and type of grinding mill employed, usually minus $\frac{3}{4}$ in. down to $\frac{3}{8}$ in.

Hardness of material will influence the selection of the design and also the type of crusher. For a very hard material, such as ferrochrome or artificial abrasives, a jaw crusher can be made to withstand the heavy impact pressures at reasonable cost better than a gyratory. For soft or nonabrasive materials, a hammer mill might work out better than either a jaw or gyratory crusher.

Moisture affects gyratory crushers as a rule more than jaw crushers. The material tends to pack between the mantle and concaves of the gyratory. Decreasing the angle of nip of a jaw crusher handling moist material helps. Where extreme conditions are encountered, a conical scrubber ahead of the crusher will wash out the sticky clay and permit the use of either a jaw or gyratory crusher. Where feed size is too large for the scrubber, a primary jaw crusher ahead of the scrubber will usually work satisfactorily. The scrubbed material is then further reduced by a gyratory or cone crusher.

Screens are of considerable aid to the crushing operation where tonnages are reasonably high. Screening ahead of a reduction crusher eliminates cushioning and interference. The capacity is increased thereby. A crusher run in closed circuit with screens need not be watched so closely to maintain size as when run in open circuit; also the capacity is greater as the opening can be increased, which increases the circulating load, and any piece that slipped through uncrushed will eventually be nipped at the minimum opening. Sizing is more uniform.

Reduction ratio affects crusher capacity, character of product, and wear rate. To produce a granular material with minimum fines, moderate ratios of reduction should be used. This may mean the addition of additional crushing stages—also with screens in closed circuit. Double-toggle Blake-type jaw crushers and rolls operate best at a moderate ratio of reduction. Single-toggle jaw crushers can operate at somewhat greater ratios to advantage, and Dodge-type jaw crushers for moderate capacities. Reduction gyratory crushers and hammer mills can operate satisfactorily with considerably greater ratios of reduction than the other types. These latter types are also employed ahead of the grinding operation. They tend to choke more with moist feed, and hence may not be satisfactory in that case; or they should be set at a larger discharge opening, primary ball or rod mills then being used and arranged to take a coarser than normal feed.

Grinding or pulverizing may be defined as the process of comminution of ore and materials by a combination of impact and attrition. It should be noted that, in this definition, the word "impact" is used, while in straight crushing operations it is also

employed. The action is much the same and differs only in degree, but in grinding there is also a secondary action—that of rubbing or attrition—which does not occur in good crushing practice. While an attritive action is not good practice in coarse grinding, it is essential when fine products are to be secured, as this seems to be the action that will produce the desired product more cheaply than straight impact.

Grinding mills are designed to encroach on the field of the reduction crusher, taking a feed as coarse as 3 in. and, in some cases, 4 in., and reducing it to a fine powder in one operation. This practice, while saving in the stages required, is by no means economical, either in power consumption or maintenance. Present practice is not to feed a grinding mill with ore or material larger than $1\frac{1}{2}$ in. maximum size, unless the operation is small and first cost is of prime importance. In large-scale operations, it has been found best practice to reduce by reduction crushers to $\frac{3}{4}$ in. and sometimes even as fine as $\frac{1}{4}$ in. before feeding to the grinding mills. Where a fine product is desired, it is also good practice to use a primary mill, followed by a regrinding mill, each designed and operated to accomplish the reduction called for.

The fundamental difference between the impact action in grinding and the same action in crushing is that, in grinding, this action is a blow similar to that of a sledge hammer sufficient to mash the piece, while in crushing it is a squeezing force under direct pressure. Theoretically, it would be better for the grinding device to crack each piece at each blow, but this is not possible or practical, owing to the nature of the mills employed or the cost of the operation. The grinding mill must effect a greater ratio of reduction than the 4 to 8:1 used as standard for crushers. For example, in primary grinding, it is customary to reduce $\frac{3}{4}$ -in. feed to all passing 10 mesh (0.065 in.). This is a reduction of nearly 12:1, and in regrinding from a feed of minus 10 mesh to minus 100 mesh (0.0058 in.), the ratio of reduction is 11:1 diameters. To effect a reduction of $\frac{3}{4}$ in. to 100 mesh in one mill is a reduction ratio of 130:1, and it is evident why such a reduction in one mill is usually uneconomical. (See notes on Grinding, page 35, for further information regarding reduction ratios and energy consumed.)

It is true, however, that when the operation is on a small scale one mill may be used to advantage to effect this large reduction, but on the whole, it is not to be considered good practice. As in crushing, the finer the product desired, the greater must be the grinding surface exposed in order to effect the necessary reduction. This is secured in the slow-speed ball or rod mill when many relatively light grinding bodies are mixed with the material to be reduced, and in the high-speed roller type of mill, where a few heavy grinding bodies revolve at very high speed, the same bodies making many repeated contacts under high pressure. In either case, a combination of crushing and rubbing or attrition exists, which, in the first action, breaks down the coarse pieces and, in the second action, grinds them into a fine powder.

In most grinding operations used in metallurgical practice, the reduction is accomplished wet, *i.e.*, with sufficient moisture to make the material fluid. When grinding dry, the moisture is never allowed to increase to the extent that the mass becomes even plastic. When grinding "wet," it is the usual practice to crush dry, then add water to the grinding mills, and carry out this reduction wet, since it has been found cheaper to grind in this manner. It does not follow, however, that it is always cheaper than when efficient air classifiers or screens are employed and the moisture is kept at a minimum or "mill drying" is practiced. In "dry" grinding, it is desirable to have the material as dry as possible.

Wet grinding is usually preferred when the subsequent process is wet, because it is a simple matter to classify the fine from the coarse particles in the discharge, and also in many operations where reagents are to assist the extraction. It has usually been found advantageous, if these reagents are added ahead of the grinding mill, because of

the intimate mix that occurs during the grinding operation. When the subsequent process is dry, then dry grinding is usually called for, since to grind wet, then thicken, filter, and dry the fine product usually entails high capital and over-all operating costs.

Open-circuit Grinding.—It was the custom a number of years ago to grind only in open circuit, *i.e.*, to pass the material through the mill once, so regulating the feed rate and other factors that the product was finished when it issued from the mill. This was, and still is, the practice in those types of mills using internal screens or sizing devices that control the size of the product and hold back the coarse until it is reduced sufficiently to pass through. Rod and tube mills are operated in open circuit owing to their lengths and also the character of the product usually wanted from these types. But with the advent of the ball and pebble mill, and better methods of screening and classifying, closed-circuit grinding has found favor. The product is better, and grinding costs are lower.

When two mills are operated in series, the first is sometimes operated in open circuit—*i.e.*, no oversize or semiground product is returned to it, but it is passed on to a second mill operating in closed circuit with a suitable sizing device.

Closed-circuit Grinding.—It has been found more satisfactory to grind so that the mill, instead of delivering a finished product on the first pass through, discharges a certain quantity of oversize. This oversize is then classified out by several well-developed methods and is either sent back to the mill for regrinding or is passed on to a second mill. In this way, not only is overgrinding and a consequent loss in energy reduced, but the capacity of a given mill is increased.

The generally accepted devices now used for classifying are screens for fairly coarse separation, and when grinding wet, some form of hydraulic classifier for finer products. For fine dry grinding, the mill is then run in closed circuit with some form of air classifier. These are described in detail later.

Types of Grinding Mills.—It is impossible, because of lack of space, to give full descriptions of the many types of grinding devices that have been used successfully in the reduction of ores and minerals. A short summary of those mills now little used will be given first, followed by a more detailed description of the mills considered as standard at the present time.

Stamps.—Stamps were considered standard practice up to as late as 1915, but after that time they were rapidly superseded by the ball mill; until now, unless the operation is of a special nature, no new installations include them. Even some of the older, but successful, installations of stamps are being gradually replaced with ball mills.

The principle of operation of the stamp (Fig. 12) is simply the mashing of the ore between a stamp shoe and a die. Water is added to wash the ground product through screens. The product that is unable to pass the screens works its way back onto the die and is finally ground so that it will pass through.

The advantage of the stamp is its ability to reduce ore as large as 4 in. to 40 mesh in one operation with a reasonable power consumption. The disadvantages are that the stamps must be carefully watched so that the shoe is not allowed to strike the die without ore between to act as a cushion. The screen wear is very high when a high ratio of reduction is to be effected. The noise made is terrific. The cost of repairs is high, and they are difficult to make, with attendant loss in operating time.

When stamps are equipped with coarse screens, from $\frac{1}{2}$ in. to 4 mesh, they make a good primary milling device, and most of the old stamp batteries in use today are so equipped. The product is then passed on to regrinding mills. This makes the stamp duty much higher and materially reduces the maintenance cost.

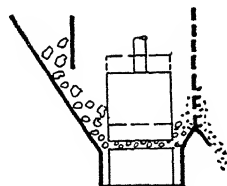


FIG. 12.—Principle of stamps.

Chilean Mills.—The Chilean mill (Fig. 13) is little used now. Several installations throughout the country are still in successful operation. The development of the conical pebble mill since 1908 is largely responsible for the change in trend.

The Chilean mill is somewhat similar to the ancient chaser mill. It consists of heavy wheels with axes horizontal, running on a die ring. The ore is fed into the center. The rolls, in running around the ring, crush the ore and splash it up against screens located around the ring. The oversize drops back on the ring for further reduction.

The illustration shows the principle of operation. The mill is primarily a regrinding mill, *i.e.*, the feed seldom exceeds $\frac{3}{8}$ in. in size, and the product 20 to 40 mesh, depending upon the subsequent metallurgical treatment.

The *advantages* of this mill are that a finished product is produced in the mill itself through the use of screens of the desired mesh, without resorting to outside classification.

The *disadvantages* are that, for the same reduction, the average power requirements are higher than for the equivalent mills now used, and the wear on rollers and dies is high. Unless kept in first-class shape they will corrugate, causing a marked decrease in grinding efficiency. Wear on the screens around the mill is severe, and they require

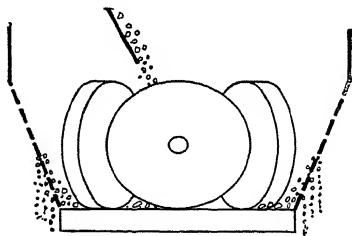


Fig. 13.—Chilean mill.

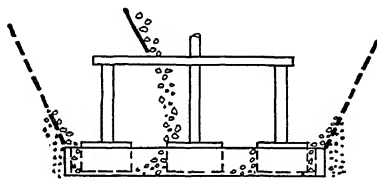


Fig. 14.—Huntington mill.

constant replacement. The loss of operating time due to shutdowns to make repairs is high.

The dry pan or chaser mill is similar in principle to the Chilean mill (Fig. 13) and is in use today mainly in the clay and ceramic industries, but usually operating as a dry-grinding unit, both as a batch and as a continuous mill. In the continuous mill, instead of having screens forming the sides, the grinding plate is perforated or a circular plate at the periphery is adjusted as to clearance, so as to allow the material to pass out. When a finer product is wanted than can be obtained by the relatively coarse perforations or clearances, then the mill is usually operated in closed circuit with a separate screen. The oversize is returned to the mill for finishing.

Advantages.—The ratio of reduction in the coarser range of sizes is large. When stone is used for the roll and plate, metal contamination is eliminated. The mill is able to crush and grind damp materials without drying. It can be used as a batch as well as continuous grinding mill.

Disadvantages.—Power in the finer size range, below about 6 to 10 mesh, is excessive. This mill is very dusty in operation. Maintenance when grinding hard materials is high. Constant attention is required.

Huntington Mill.—The Huntington (wet-grinding) mill (Fig. 14) has been little used for a number of years. It operates on the principle of rollers, their axes nearly perpendicular, being pressed against a die ring by centrifugal force, as the arms are rapidly revolved. The ore, as in the Chilean mill, is crushed between the roll and the ring. The product is splashed against screens around the ring, the oversize slides

down the screen and returns to the ring to be further ground. The sketch shows the principle of operation.

The advantages of the Huntington mill are similar to those of the Chilean mill. The disadvantages are the same as those of the Chilean mill, with greater emphasis on the repair and maintenance costs. The Huntington mill also runs at high speed. If it had not been for mechanical troubles, probably neither the Chilean nor the Huntington mills would have lost their popularity as rapidly as they have. The basic principle as a dry mill, however, is extensively used today on relatively soft non-abrasive materials as described later on.

Buhrstone Mills.—The buhrstone mill (Fig. 15) is one of the oldest forms of grinding mills. It was used as far back as history records, to pulverize grains as well as hard materials. In principle, the mill consists of two flat stones, of which one is caused to revolve. The feed is introduced at the center of one stone and works its way to the periphery, due to grooves in the stone so cut as to push the material outward as it is being ground between the two surfaces. Fineness is controlled by the pressure exerted and by the speed. The illustration shows the horizontal buhrstone.

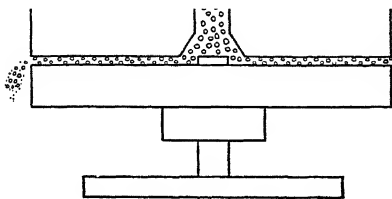


FIG. 15.—Buhrstones.

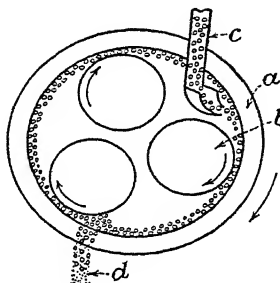


FIG. 16.—Ring roll mill.

Another form is to place the stones vertically (horizontal axis), which causes the material to work through by gravity as well.

This mill is being used less and less today, particularly for grinding hard or abrasive materials. The capacity is low, and the power per ton is high. Maintenance, when grinding anything but soft material or grain, is excessive.

Fine-grinding Rolls.—Until the advent of the ball and tube mill, rolls were often used to obtain a product as fine as 30 mesh. The advantage of the roll as a medium-fine grinder is its ability to deliver a very uniform product.

The great disadvantage of the roll as a fine grinder is the excessive maintenance cost, as the surfaces must be kept in first-class shape for fine grinding. Otherwise, the efficiency drops materially. Fine grinding rolls must also be operated in closed circuit with screens, which means the use of an elevator in the circuit.

Ring Roll Mill.—A modified form of roll is the ring roll mill (Fig. 16) designed to secure a finer product than can economically be obtained by the standard form of reduction rolls. Two designs are employed. The Sturtevant ring roll mill rotates the large ring *a* with concave inner surface through a cantilever spider attached to the ring driven through a horizontal shaft. The rolls *b* with convex faces are mounted on a hinged housing, also cantilevered through the housing and held against the ring by heavy springs. The feed *c* is introduced in front of one of the rolls, and that which is not pushed off clings by centrifugal force to the concaved inner face of the ring, since the ring is rotated well above the speed needed to cause centrifugal cling. The

material is pushed off in front of each roll, as indicated in one case as at *d*, in a semiground state. From a hopper, not shown, this semiground material is then conveyed and elevated to a suitable sizing device, either a screen or air classifier. The oversize is then returned to the mill with the original feed.

The Kent Maxecon mill has a concave ring and convex rolls, as illustrated, but the power is imparted to one of the rolls instead of the ring. The friction between the roll, material, and ring causes the ring to rotate, which also drives the other rolls, and reduction is completed in a similar manner to that described above.

Advantages.—The mills are satisfactory in grinding dry, to moderate finenesses, relatively soft nonabrasive materials, and in this range the power and maintenance are reasonable. The first cost is also reasonable, and the mill requires relatively low floor space.

Disadvantages.—Power per ton and maintenance become excessive when used to grind fine, even on relatively nonabrasive materials. Since separate screens or air classifying systems are required and the circulating loads required to maintain efficient operation are high, the simplicity of the mill itself is largely nullified by the auxiliaries required. The mill is unsuited to grind hard or abrasive materials as the maintenance becomes excessive. Power and circulating loads become excessive when used as a fine grinding unit, even on soft materials.

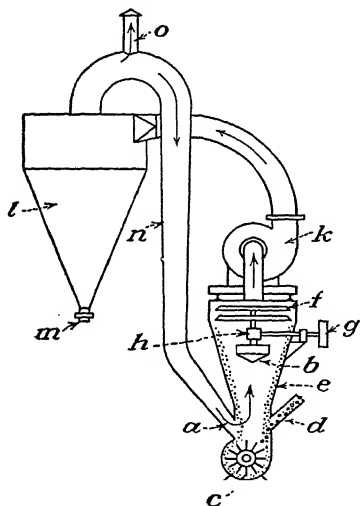


FIG. 17.—Hammer mill with air classification.

to the product collector *l*, and the finished product is separated from the air and delivered at the discharge spout *m*. The material-free air returns to the mill through pipe (*n*). Infiltration air and moisture are discharged through vent *o* to the atmosphere or, if too dusty, to a bag or other type of high-efficiency dust collector.

By this arrangement, fine products can be obtained, much finer than by the screen-type hammer mill. This system is extensively used in pulverizing hydrated lime and other soft nonabrasive materials, usually to produce finenesses ranging from 100 to 325 mesh.

Advantages.—The system is low in first cost, and dustless. The product is delivered to any reasonable height above the floor without auxiliary conveyers or elevators. When auxiliary hot air is used, damp materials can be ground satisfactorily. Fineness

Impact Hammer Mill.—A modified form of the hammer mill (Fig. 9) is the arrangement shown in Fig. 17. Instead of the feed *d* after being crushed by hammers *c* discharging through a screen grate in the bottom of the mill, air enters at *a* and carries with it the ground and semiground product into chamber *b*. The oversize slides back along the housing *e* back into the hammers *c* for regrinding. The finished and semi-ground material as it rises in the chamber encounters revolving single or double blades *f*, which cause a rapid cyclonic motion to the air and material in the upper portion of the chamber and throw out the coarser particles against the wall when they slide back into the hammer mill. The fineness is controlled by the speed of these blades, which are actuated by outside motive force through pulley *g* and transmission *h*. Fan *k* draws the material-laden air from the mill and delivers it

can be controlled within reasonably close limits, and extremely fine products are attainable.

Disadvantages.—Only soft nonabrasive materials can be ground without excessive maintenance. Frequent overhauling of the hammer mill is required. It is unsuited for use on materials that tend to reduce to a unit grain size, as this size builds up and reduces the capacity appreciably.

Roller Mills.—Figure 18 diagrammatically illustrates the principle of the roller mill, as used for dry grinding. The operation is similar to the Huntington wet-grinding mill (Fig. 14) except that this mill is essentially a dry-grinding device. Instead of classifying by screens, air is used to scavenge the mill of fines, and, as the air rises, it carries with it the finely ground material with some oversize. The air system is similar to that described above for the impact hammer mill (Fig. 17). In one design where close regulation of fineness and extreme fines are not required, 60 to 80 per cent minus 200 mesh, the classification means is simply the space above the rollers and centrifugal motion imparted to the air by the revolving rollers and arms. The coarser particles settle back along the sides into the grinding zone for regrinding. For closer control of fines and particularly where fine products from, say, 80 per cent minus 200 mesh to 99 per cent minus 325 mesh are desired, the type of classifier with auxiliary blades, as shown in Fig. 17, is employed. Table 8 covers data on Raymond low- and high-side roller mills, with capacities, grinding bituminous coal of good grindability.

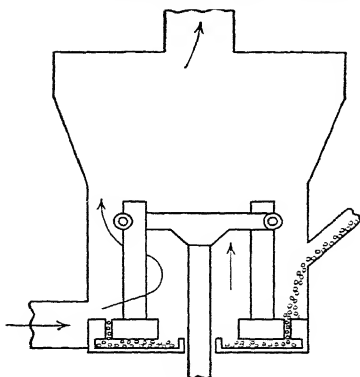


FIG. 18.—Roller mill.

TABLE 8.—DATA ON RAYMOND ROLLER MILLS¹

Size of mill	Type	Product Fineness Coal	Capacity, lb. per hr. bituminous coal	Power	
				Mill motor hp.	Fan motor hp.
3,036	Low side	70% - 200 mesh	3,200	25	15
3,036	High side	85% - 200 mesh	2,100	20	15
4,237	Low side	70% - 200 mesh	6,500	50	30
4,237	High side	85% - 200 mesh	4,400	40	30
5,448	Low side	70% - 200 mesh	13,500	100	50
5,448	High side	85% - 200 mesh	9,000	100	50
6,669	Low side	70% - 200 mesh	30,500	200	100
6,669	High side	85% - 200 mesh	20,500	200	100
73,612	Low side	70% - 200 mesh	50,000	300	150
73,612	High side	85% - 200 mesh	33,500	250	150

¹ From Raymond Pulv. Div., Combustion Eng. Co.

The Bradley Hercules mill is a roller mill, very similar to the Huntington mill, but grinds dry and uses internal screens. No air is employed, but the fines through the screens are discharged into a conveyer and used as is or sent to a regrinding mill, since

this arrangement is seldom equipped with screens finer than 10 to 12 mesh, as the screen wear and lost time, due to making screen repairs, becomes excessive. Also, unless the material is thoroughly dried, the screens blind.

Advantages.—Roller mills will grind harder materials than impact hammer mills. Critical size does not accumulate when grinding those materials that cause this difficulty in impact hammer mills. The same other advantages as mentioned for the impact hammer mill apply here, and mills of high capacity (and power) can be built without unduly complicating the mechanical arrangement.

Disadvantages.—The roller mill, like similar high-speed mills, can grind only relatively soft and nonabrasive materials economically. Repairs are frequent, oil and grease consumption is appreciable, and skilled labor is required to keep properly adjusted. Capacity and efficiency is at a maximum when ring and rollers are new

but decreases rapidly as these parts begin to wear, since wear is seldom uniform, owing to the tendency for both rings and rolls to corrugate.

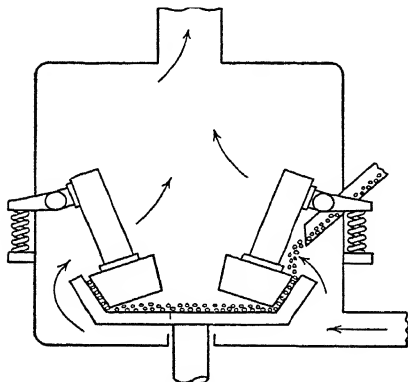


Fig. 19.—Bowl mill.

Bowl Mills.—Instead of the rollers revolving around a stationary ring, as with the roller mill (Fig. 18), the ring in the form of a bowl rotates about stationary rollers, as illustrated by Fig. 19. The stationary rollers are mounted on a pivoted yoke and are held down by strong external springs, which only come into play if the mill is overloaded, or uncrushable material like tramp iron finds its way into the grinding zone. The mill is scavenged by air, and the same system of collection and air circulation is employed as described for the impact hammer mill (Fig. 17). It is also extensively

used as a unit coal pulverizer, in which event hot air is introduced to dry and grind the coal that is removed by this air and blown directly into the furnaces, as illustrated and described in more detail later (see Fig. 34).

The Raymond bowl mill is primarily used to grind bituminous coal, as well as other relatively soft nonabrasive materials pulverized to moderate finenesses in this mill. As a result, pulverizer sizes range from units with capacities of 1 to 18 tons per hr., with powers of 40 to 300 hp. and drive shaft speeds of 1800 and 900 r.p.m., respectively.

Advantages.—It is quiet, compact, smoothly operating unit, requiring small floor space for the capacity attainable. Greasing and adjustment can be done without shutting down the mill. The mill is economical in power consumption when grinding to moderate finenesses.

Disadvantages.—Common with all high-speed mills, the bowl mill can grind only relatively soft nonabrasive materials without excessive maintenance, and efficiency is at a maximum when bowl ring and rollers are new. Is not so suitable for very fine grinding as other forms of roller and ball mills.

Ball-bearing Mills.—Figure 20 illustrates the ball-bearing principle of grinding in one of its preferred forms. A row of closely spaced balls *a* is rotated by the rotating race *b* on a heavy frame *c* by means of shaft *d* driven by gear *e*. The balls roll on race *b*, owing to contact under pressure with upper race *f*, through adjustable springs *g*. Feed enters through chute *h* and into the lower race when the balls under heavy pres-

sure crush the material as they roll. Air through duct *j* sweeps the pulverized material up into *k*, where it is classified by baffles and centrifugal motion. The fines pass out through pipe *l*. The balance of the system is similar to that illustrated in Fig. 17, for a storage system, in which the fine product is separated from the air and the air is returned to the mill, or as in Fig. 34 as a unit pulverizer, where hot air is used and the product blown directly to a furnace. For higher capacities and finer products, three sets of balls and races are incorporated in one mill. This is known as the type B Babcock and Wilcox ball-bearing mill. One large set of balls receives the coarse feed

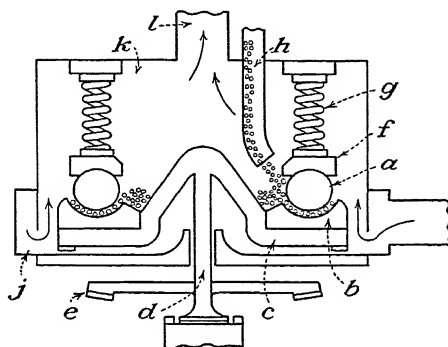


FIG. 20.—Ball-bearing type mill.

first, and as the material passes through the mill it is directed into an outer row of smaller balls and then to the third row below the first two. The mill is either air swept, or the product is allowed to drop out at the bottom, where it is conveyed to a separate classifier, similar to the arrangement illustrated in Fig. 32 for a ball mill.

Table 9 covers data on various sizes of type B Babcock and Wilcox pulverizers grinding limestone.

TABLE 9.—BALL-BEARING TYPE B BABCOCK AND WILCOX MILL DATA

Size of mill	Hp. required	Capacity on limestone, tons per hour		
		To 90% — 200 mesh		To 96% — 200 mesh
		Medium hard	Medium	Medium
220	117	5.5	7	6
226	180	8.3	12	10
238	270	12.5	17	15
346	410	18.7	26	22
360	560	26.0	36	30
366	680	31.0	43	36

The Fuller mill is another and earlier form of ball-bearing pulverizer, differing from the above by using larger balls, pushed around a race by pusher arms instead of utilizing two races. A screen around the periphery of the mill is used to size the

product or air, similar to the method described for the Babcock and Wilcox air-swept mill.

Advantages.—The mill is compact and relatively low in power consumption on soft materials. It requires little floor space. Greasing and adjustments are made from the outside of the mill.

Disadvantages.—It is suited to grinding only relatively soft, nonabrasive materials if maintenance is to be kept at a reasonable cost. Practically complete dismantling is required to renew worn or broken races. Efficiency drops with use, as grinding surface becomes uneven or the total area of contact is reduced as the balls become smaller and the races change in contour. When outside classification is employed, circulating loads are extremely high, thus adding to auxiliaries and over-all power consumption.

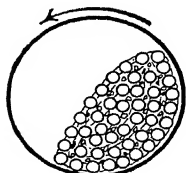


FIG. 21.—Diagrammatic section of tube mill.

Rotating-drum Mills.—In this category are the most universally used of all modern grinding mills. The mill is essentially a slowly rotating drum on a horizontal axis, partially filled with freely moving grinding bodies, harder and tougher than the material to be ground. The tumbling action of these bodies crushes and grinds the

material as it is fed to and passes through the mill, if the operation is continuous. If charged into and remaining in the mill during operation, the operation is in batches. Figure 21 illustrates the basic principle.

Mills of this type fall into three general classifications, aside from being continuous or batch in operation.

1. Ball mills using metal balls.
2. Pebble mills using a nonmetallic grinding media, such as flint pebbles or larger pieces of the material itself, provided that it is hard and relatively tough.
3. Rod mills that use rods the full length of the mill, that freely tumble like balls or pebbles as the mill rotates.

These mills are essentially slow-speed grinding devices, 12 to 40 r.p.m., depending upon the size. The slow speed, together with the inherent simplicity and ability to run continuously over long periods and under adverse conditions, is the chief reason why this type of mill is so generally employed for wet and dry coarse and fine grinding. It is the only type of mill, in certain improved forms, that can grind very hard and abrasive materials at a reasonable cost.

Krupp Mill.—As a preliminary grinder, the Krupp ball mill (Fig. 22) was long considered standard. It is still used today but is gradually giving way to the types of mills later described. Material is fed into the center of the mill *a*; balls in dropping and rolling down the cascade liners *b* crush and grind the coarser pieces. As the drum rotates, the ground and semi-ground product drops through holes *c* to coarse screen plate *d*. Fines that will pass the screen plate drop on to fine screen *e*. The undersize of this screen drops out of the rotating mill and into stationary hopper *f*, which also is part of a cover over the entire mill, to prevent dust from escaping. The oversize of both coarse screen *d* and fine screen *e* join at *g* and pass back into the mill in a space

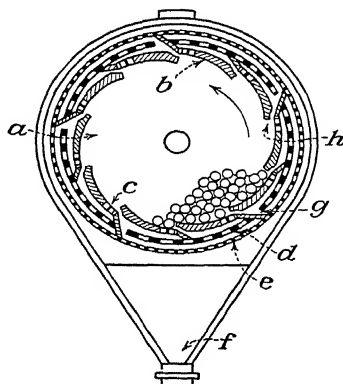


FIG. 22.—Krupp mill.

under liner plate at h for finishing. The product obtained is granular and devoid of

Advantages.—The power per ton ground through a given mesh size is good, but since few fines are produced the total work done does not compare favorably with mills of later design.

Disadvantages.—The maintenance, particularly on the screens, is high, and the mill requires dismantling to repair. Even though a dust hood surrounds the mill, considerable dust escapes through cracks, unless constantly watched and kept tight. It is suitable only for relatively coarse grinding operations.

Cylindrical Ball Mills.—Figure 23a illustrates the standard trunnion-overflow or center-discharge cylindrical ball mill. The mill consists primarily of a drum rotating about its horizontal axis on hollow trunnions. The drum is driven by means of a circumferential gear, through pinion, on a separate countershaft, which in turn is driven by belt-reduction gearing or direct connected through flexible coupling to a motor. Feed is introduced, through the feed-end trunnion, by a scoop attached to, and rotating with, the drum. The mill is filled approximately half-full of balls, which cascade or tumble down the incline formed as the drum is rotated. Owing to the difference in head caused by the feeder, the material being ground migrates toward the discharge end, where it flows out of the hollow discharge trunnion in a ground or

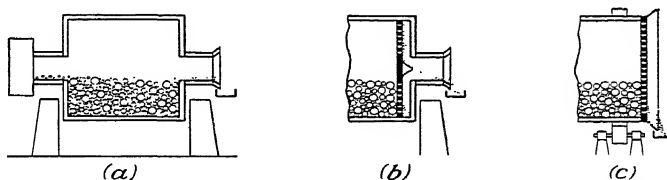


FIG. 23.—Cylindrical mill: a, trunnion overflow; b, grate discharge; c, peripheral grate discharge.

semiground state, depending upon the rate of travel through the mill. The higher the feed rate, the greater the difference in head and the faster will be the travel toward the discharge end, so long as the material will discharge as fast as it is being fed.

The mill is adapted and used for both wet- and dry-grinding operations. When grinding wet, the rate of travel is considerably faster than when grinding dry, as less head is required to move a given material volume through the mill under the same conditions, thus making this form of mill more flexible and able to grind to better advantage wet than dry. Fineness of product and capacity are dependent upon speed, ball size, loading, and ratios of different ball sizes, size of feed, hardness of material being ground, moisture content, and control of feed rate.

Advantages.—The mill is of simple construction and readily inspected and repaired. If crowded too much and overloaded, it will relieve itself much more readily than those types with grate or restricted-discharge arrangements. Maintenance is but a fraction of that of high-speed mills, grinding either wet or dry. The wet-grinding with medium size feed is capable of maintaining very high circulating loads.

Disadvantages.—Power consumption and maintenance are somewhat high with certain other forms of trunnion-overflow mills grinding wet and consume more when grinding dry. When fed coarse material, it tends to discharge trash size, as there are no means to hold back larger pieces that short-circuit the mill.

Grate-discharge Ball Mills.—There are two general types of grate mills; the type with internal grate and lifters (Fig. 23b) and the open

23c). The object of using a grate is to lower the material or pulp level so as to increase the rate of migration through the mill and to hold back tramp oversize from discharging with the product.

The internal-grate-discharge mill (Fig. 23b) operates in the same manner as the trunnion-overflow mill described above, except for the grate discharge. The grate is made of a thick casting or built-up alloy steel bars closely spaced and tapered outwardly, to prevent lodgment of coarse pieces and tramp iron between the openings. Between the grate and end wall are lifters that, as the mill rotates, pick up the product that has passed through the grates and discharge it into the discharge trunnion. A coarse grate permits rapid flow of the material through the mill but also permits tramp oversize to pass through, as well as small balls that may be large enough to be effective. Grate openings are, therefore, specified to suit conditions and vary from a minimum opening of about $\frac{3}{16}$ in. to a maximum of about 1 in.

The open end mill (Fig. 23c), instead of being supported on trunnion bearings, is equipped with a tire around the shell which is supported on rollers. The product from the grate discharges by gravity peripherally, the object being to attain full diameter discharge and a minimum of material loading in the mill.

The Marcy mill is a grate-type mill, built with two types of grates (Fig. 23b and 23c). Three methods of support are employed: two trunnions (Fig. 23a), trunnion at feed end and tire at discharge end (Fig. 23a and c), or tires at both feed and discharge ends. Designation of size, unlike other makes, is measured as the volume inside of the new liners and length between feed head liners and grate. Sizes range from 30 in. diameter by 36 in. long to 10 ft. diameter by 10 ft. long, mill ball loads from 1900 to 113,000 lb., and power from 5 to 750 hp. Capacity ratings for wet-grinding mills with a feed of $\frac{1}{2}$ in. and grinding to 65 mesh in closed circuit with a suitable classifier range from 8 to 1700 tons per 24 hr.

Sizes, power, and approximate capacities of Allis-Chalmers mills are given in Table 10. This is typical data for both grated-type and trunnion-overflow mills in various sizes for grinding wet. Traylor Engineering & Mfg. Co., in its capacity tables, states that dry-grinding cylindrical mills have about 60 per cent of the capacity of wet-grinding cylindrical mills to the same fineness.

Advantages.—Grate-discharge mills can grind a coarser feed size than the equivalent trunnion-overflow mill without discharging tramp oversize with the product. When used for grinding materials dry, the rate of travel through the mill is increased over the trunnion overflow type, with less tendency to cushion the balls. A grate installed in a wet-grinding trunnion-overflow mill will increase the capacity of the mill but with an increase in power consumption at least equal to, and usually somewhat greater than, the equivalent increase in capacity. On coarse feeds, a relatively high circulating load can be maintained without discharging tramp oversize.

Disadvantages.—Compared with the equivalent trunnion-overflow mill, grate mills operate with higher over-all lining and ball consumption, in addition to grate maintenance. The wear is due to greater opportunity of metal-to-metal contact. Grate openings tend to plug up with broken balls, tramp iron, and wood chips. An overloaded grate-type mill usually requires several hours to clear up, compared with a few minutes for the trunnion-overflow mill. The peripheral-discharge grate mill requires an auxiliary pump to elevate the product to its classifier when operated in closed circuit. Wear is usually greatest at the discharge end owing to lack of material there and the tendency of the large balls to seek that zone which has greatest free motion. The over-all work done (mesh tons per horsepower) is no better than a trunnion overflow mill taking the same power.

Conical Ball Mill.—The Hardinge Conical ball mill (Fig. 24), like the cylindrical mill, is a drum rotating about its horizontal axis and operated in much the same way.

CRUSHING AND GRINDING

TABLE 10.—CYLINDRICAL BALL MILL CAPACITIES¹

Size, ft.	45 % ball load, lb.	R.p.m.	Short tons per 24 hr.		Motor output, hp.	Motor size, hp.
			65 % — 200	85 % — 200		
3 × 5	3,700	44	33	23	23	25
4 × 4	5,600	37½	50	35	34	40
5 × 5	11,000	30½	100	70	68	75
6 × 6	19,500	26½	180	125	120	125
6½ × 8	30,500	25	290	205	187	200
7 × 8	35,500	23½	335	235	216	225
8 × 8	47,000	21½	460	325	290	300
9 × 9	68,000	19½	685	480	420	425
9½ × 10	83,000	18½	830	575	500	500
9½ × 12	99,500	18½	1000	695	600	600
10½ × 12	123,000	17	1275	900	750	750

¹ From Allis-Chalmers Mfg. Co. *Bulletin*.

Data based on wet grinding; single stage; closed-circuit operating; grated-type mills, feed (½ in. one-way dimensions); "diameters," inside shell without liners; "lengths," working length shell between end liners.

For trunnion-overflow mills, reduce both power and capacity approximately 20 per cent.

Unlike the cylindrical mill, the ends are conical. The feed-end cone is of greater inclination than the discharge end, as shown by Fig. 24*a*, or as in the Tricone mill the central portion is a truncated cone (Fig. 24*c*). The conical shape accomplishes three main objectives: (1) creates a ball action that roughly proportions the energy to the work done; (2) increases circulating action and rate of migration through the mill; (3) decreases ball, liner, and, if used, grate wear.

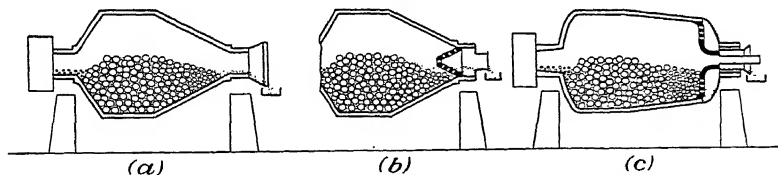


Fig. 24.—Hardinge Conical mill: *a*, high pulp level discharge; *b*, reverse cone grate; *c*, Tricone with grate.

Taking advantage of the phenomenon that a large lump will roll downhill ahead of a smaller one, the cone deflects the larger freer moving balls to the feed end (largest diameter) of the mill where the largest size particles are encountered. The smaller balls are displaced and congregate in a zone of lesser diameter which is the discharge cone. The material, in passing through the mill and being broken into small particles by the larger balls, is more easily and efficiently ground by the smaller balls, which have greater surface area per unit of volume and fall from a lesser height, and at lower peripheral speeds.

The tapered or conical end, on raising the material against the shell and lifters above the center line, causes it to fall forward and downward, not merely downward, with the effect of rapidly moving the material toward the discharge end, at a much

greater rate than by displacement only. The skin friction against the liner and lifters is similar to having an infinite number of small inclined chutes continuously shoving the material forward. The finer particles with greater surface area and frictional resistance are influenced more than the coarser pieces and will be propelled forward more rapidly. The larger pieces will remain in the large diameter zones until broken down.

Wear on balls and liners is lower where the force exerted can be proportioned to the work required. It follows that ball and liner wear in the conical ends is lower with consequent over-all reduction in wear per ton of product. When a grate is used in the discharge cone, its wear is reduced since only the smaller balls come in contact with it. Grates are employed in Conical mills to keep back tramp oversize if the feed is coarse, but need not be employed to increase the circulating load as the forward propelling action of the cone will keep the flow rate up to any practical limits of good classification. Table 11 covers sizes and capacities of Hardinge Conical ball mills.

Figure 24a shows the appearance of balls inside the mill. The large balls remain in the central portion, and the size decreases as the discharge end is approached. The trunnion opening shown in this illustration is the ultra-high pulp level for wet grinding. Raising the over-all mill load and moment of inertia increases the grinding rate without an equivalent increase in power. Figure 24b illustrates the arrangement with reverse cone grate to hold balls back and keep the grate from blinding by the cleaning action that occurs with this arrangement.

The Hardinge Tricone mill is illustrated by Fig. 24c, a development of the Conical-mill principle that aligns the balls as mentioned above but provides a larger space for

TABLE 11.—CONICAL BALL MILLS¹

Size of mill	Maximum weight of balls lb.	R.p.m. of mill		Hp. of motor		Capacity, tons per 24 hr.				
		Min.	Max.	Min.	Max.	1½ in. to 10 mesh	¾ in. to 48 mesh	½ in. to 100 mesh	¼ in. to 98 mesh	¼ in. to 98 mesh
2' × 8"	650	35.2	46	1.5	2	0.7	3.5	2.5	1.5	1
3' × 24"	2,400	30.4	39.8	10	10	32	16	12	7	4
4½' × 16"	4,600	25.2	33	15	20	65	33	23	15	9
5' × 36"	11,000	23.2	30.4	40	50	174	88	61	39	23
6' × 36"	17,500	21.2	27.7	60	75	282	140	97	64	38
7' × 36"	26,000	19.4	25.4	100	125	465	225	155	102	61
8' × 48"	43,500	18.2	23.8	175	200	820	394	274	179	108
9' × 48"	58,000	17.3	22.4	250	300	1290	630	435	285	172
10' × 48"	67,500	16.2	21.2	300	400	1690	810	560	367	221
10' × 72"	88,000	16.2	21.2	400	500	2040	970	675	440	266
12' × 72"	136,000	14.7	19.2	700	800	3300	1570	1090	715	430
12' × 120"	196,000	14.7	19.2	1100	1200	4900	2340	1630	1060	640

¹ From Hardinge Co., Inc., *Bulletin*.

Capacities given are for trunnion-overflow mills grinding material wet of average grindability. For hard materials, the capacities should be reduced, and for softer materials, increased. Dry-grinding capacities are 80 to 90 per cent of the above capacities. Normal operating ball charges for mill with air classifiers are 15 per cent below those listed.

the smaller balls, thus increasing the grinding action on the semiground material. The illustration shows a grate that is employed in this arrangement when conditions warrant, but being where the small balls only are in action, the wear is relatively light.

Advantages.—The Conical mill is low in power consumption for a given reduction, is simple to operate, and, if overloaded, can be brought back to normal in a very short time owing to its free discharge and rapid forward propelling action. The use of a grate is unnecessary to secure maximum capacity under normal circumstances. The shape permits the making of a structure of great strength with a minimum of weight. Bearing friction and power wastage are thus reduced to a minimum.

Disadvantages.—For grinding soft nonabrasive materials dry, the Conical mill is not so low in power consumption as some high-speed mills. It requires more floor space than the equivalent high-speed mill. When grates are used, the same disadvantages mentioned for the grated-type cylindrical mill hold true here except that grate wear is not so high since only the smaller size balls are in contact with the grate. Maximum efficiency is not secured unless operated in closed circuit with some sizing device such as a screen or classifier unless the range of reduction is relatively low.

Pebble Mills.—The pebble mill, either Conical or cylindrical, operates on the same principles as the ball mill, but since flint pebbles or hard selected larger pieces of the material being ground are used, the capacity for a given grinding volume is considerably less. The power required is also less. The efficiency may be comparable with the ball mill in many instances. Feed size of a cylindrical pebble mill should seldom be over 6 mesh, and for the Conical pebble mill $\frac{3}{8}$ to $\frac{1}{2}$ in., otherwise even with large pebbles up to 5 or 6 in. in diameter the mill may not operate efficiently. The Conical mill is used without a grate when grinding dry except under special conditions. A

TABLE 12.—CONICAL PEBBLE MILLS¹

Size of mill	Weight of pebbles		R.p.m. of mill	Hp. of motor		Capacity, tons per 24 hr.				
						$\frac{1}{2}$ in. to 20 mesh	$\frac{1}{2}$ in. to 65 mesh	$\frac{1}{2}$ in. to 100 mesh	$\frac{1}{4}$ in. to 98% -200 mesh	$\frac{1}{4}$ in. to 98% -325 mesh
	Min.	Max.		Min.	Max.					
2' × 8"	80	100	51.5	0.5	0.1	0.2	0.9	0.75	0.5	0.275
3' × 24"	550	700	40.4	5	5	15	7	5.5	3.5	2.1
4½' × 16"	1,150	1,500	33	7.5	10	34	15	12	8	4.5
5' × 36"	2,400	3,100	31.2	15	20	72	32	26	17	10
6' × 36"	3,800	4,800	28.3	25	30	117	52	42	27	17
7' × 36"	5,800	7,500	26	40	50	186	83	66	43	26
8' × 48"	9,900	12,700	24.1	60	75	326	146	117	76	45
9' × 48"	13,500	17,400	22.6	100	100	455	204	163	106	64
10' × 48"	15,900	20,400	21.4	100	125	540	242	194	125	76
10' × 72"	20,800	26,800	21.4	125	150	710	318	255	165	100
12' × 72"	32,800	42,500	19.4	200	250	1130	505	402	262	157
12' × 120"	47,600	61,200	19.4	300	400	1620	725	580	376	226

¹ From Hardinge Co., Inc., *Bulletin*.

Capacities given are for grinding material wet of average grindability. For hard materials, the capacities should be reduced, and for softer materials, increased. Dry-grinding capacities are 80 to 90 per cent of the above capacities. Normal operating pebble charges for a mill with air classifier are 5 to 10 per cent below minimum listed.

cylindrical pebble mill without a grate is quite sluggish in operation; when a grate is used, pebble breakage is high and metal contamination of the product may occur. Table 12 gives data on Conical pebble mills, and Table 13 on cylindrical pebble tube mills.

Advantages.—Pebble mills, using a nonmetallic lining like silex or porcelain, grind without contaminating the product with iron, a very important consideration in the ceramic industry and in handling certain chemical products. On certain materials, over-all grinding costs including lining and grinding media are less than the equivalent ball mill or high-speed pulverizer. When so adjusted, a Conical pebble mill can produce a more uniform product than any type of ball mill.

Disadvantages.—Capacity for a given mill volume or capital outlay is usually less than for the equivalent ball or high-speed mill. The mill will not handle as coarse feeds so satisfactorily as a ball mill or some high-speed mills.

TABLE 13.—PEBBLE TUBE MILL DATA^a
Trunnion Overflow

Size, diam. and length, ft.	Weight of 4-in. quartzite lining	Weight of 1½-in. metal lining	Normal pebble charge, lb.	Hp. of motor	Normal speed, ^d r.p.m.
3 × 6	3,000 ^b	4,800	1,500	10	38
4 × 10	8,200	9,300	3,900	25	33
5 × 16	15,700	17,500	13,200	50	29
5 × 22	20,800	23,500	18,000	75	29
6 × 20	19,000	21,000	23,300	100	26
7 × 22	24,700	28,000	35,200	150	24
8 × 24	32,000	36,500	43,500	200	22
9 × 24	38,100	52,000	59,000	250	20

^a From Hardinge Co., Inc., *Bulletin*.

^b 2½ in. Quartzite lining.

^c With grate, pebble loads can be increased as much as 15 per cent. Power increases proportionately.

^d Speeds are for quartzite lining and wet grinding. For metal wave type, decrease speeds 7½ per cent. For dry grinding, decrease mill speeds 10 per cent.

Ball-tube and Compartment Mills.—The ball-tube mill is a long cylindrical mill using small balls. Originally it was operated in open circuit. To obtain a fine product in one pass, the barrel was made relatively long compared with the diameter so as to ensure a product with a minimum of oversize. Much overgrinding was done. Today this mill is used mainly where a product containing an excess of fines with some coarser sizes mixed in is desired, or where a plastic or smooth character of product is required. The mill is not so efficient in meeting size specifications as shorter mills operated in closed circuit with some suitable sizing device. Feed size seldom exceeds 6 mesh unless the material is soft.

The compartment mill (Fig. 25) is a ball-tube mill with compartments to keep the balls zoned into various major sizes as illustrated. By this means, the mill can then be fed with sizes as coarse as 2 in. and deliver a fine product without the use of an outside classifier. The system is compact and relatively simple but not efficient.

The mill was operated in open circuit in the manufacture of cement, until specifications became more strict and fineness requirements increased. Now it is generally closed circuited with a suitable air classifier with a material increase in over-all

efficiency. Sizes range from 5 ft. in diameter by 22 ft. long, requiring about 200 hp., to 10 ft. in diameter by 30 ft. long, requiring up to 1500 hp.

Rod Mills.—The rod mill is similar in appearance and in principle of operation to the ball mill except that rods are employed as grinding media in place of balls or pebbles. The rod mill, after a number of years of misapplication, is now generally used in a rather narrow or special field of reduction, somewhere between a reduction gyratory crusher or roll and a ball mill. It is particularly suited to grinding in open circuit a feed size up to about 1 in. and delivering a product of about 10 mesh, either wet or dry. The mill is used for finer products, but the over-all efficiency suffers as

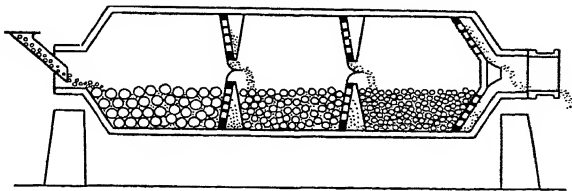


FIG. 25.—Compartment mill.

compared with the equivalent ball mill. Where both mills are operated in open circuit, a rod mill produces a minimum of tramp oversize for a given reduction. The ball mill, to eliminate tramp in one pass, must be underfed and tends to overgrind. On the other hand, a ball mill, if run in closed circuit with screens or efficient classifiers, will often do an even better job in the same range.

Figure 26 illustrates several designs of rod mills. Figure 26*a* is the trunnion-overflow vertical-ended rod mill as employed for wet-grinding operations and is similar in design to the standard cylindrical ball mill.

Figure 26*b* shows the standard Marcy rod mill with large discharge opening and splash door mounted on a separate pedestal and capable of being swung back for

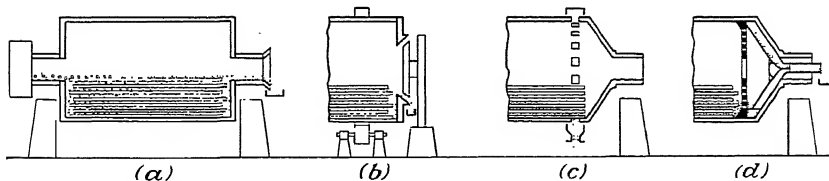


FIG. 26.—*a*, Rod mill with trunnion overflow discharge; *b*, Marcy open-end mill; *c*, Hardinge peripheral discharge with slots; *d*, same with lifters.

inspection or replenishment of rods. This mill is tire mounted and runs on rollers similar to the ball mill shown in Fig. 23*c*.

Figure 26*c* and *d* illustrate two arrangements of the Hardinge rod mill with conical ends at both feed and discharge. Rods align themselves vertically even though the ends are conical. Any tendency of a rod to move laterally is soon corrected by the rod end in rolling down the incline contacting the sloping head which causes the rod to be deflected back into alignment. The space at the feed end affords ready access of feed between the rods, and at the discharge end tends to throw back any tramp oversize that may have gotten through. Figure 26*c* is a peripheral discharge arrangement particularly suited to dry grinding as it affords maximum difference in head between feed and discharge, which increases the rate of flow; an important considera-

tion in those cases where damp materials are being ground. The Tompkins Mining Machinery Co. also makes a mill of this type. Figure 26*d* is another low pulp-level arrangement to obtain a low-level discharge yet elevate the pulp to a point near the center line where it is then possible to run in closed circuit with a classifier without an auxiliary pump.

Table 14 gives data on Hardinge wet- and dry-grinding rod mills. Table 15 gives the Marcy rod-mill catalogue capacity estimates when grinding 1-in. medium ore feed to the finenesses indicated. Capacities may vary considerably over or under those indicated, depending upon operating conditions and hardness of the ore being treated.

Advantages.—The rod mill is a very satisfactory reduction crusher and in its field operates with less attention and maintenance than equivalent reduction gyratory or

TABLE 14.—ROD MILL DATA¹

Size of mill, ft.	Weight lb. rod charge	R.p.m. mill		Horsepower				Capacity	
				To run		Motor		Tons per 24 hr. wet grinding $\frac{3}{4}$ in. to 20 mesh	Tons per hr. dry grinding $\frac{3}{4}$ in. to 10 mesh
		Min.	Max.						
2 × 4	1,700	32	43	5	6	5	7.5	27	1
3 × 6	6,000	25	34	15	18	15	20	80	3
4 × 8	14,000	21	29	33	40	35	40	180	6
5 × 10	27,000	18	25	60	80	75	100	350	12
6 × 12	50,000	16	19	110	140	125	150	620	22
7 × 14	80,000	14	17	175	220	200	250	970	34
8 × 14	105,000	12	15	225	280	250	300	1250	44
9 × 14	135,000	11	13.5	280	350	300	350	1550	55

¹ From Hardinge Co., Inc., *Bulletin*.

TABLE 15.—APPROXIMATE CAPACITIES OF ROD MILLS¹
Based on Medium Hard Ore—1-in. Feed

Size of mill, ft.	Tons per 24 hr.				
	8 mesh	20 mesh	35 mesh	48 mesh	65 mesh
2 × 4	20	15	12	10	7
3 × 6	105	80	65	50	40
4 × 8	240	180	145	120	90
5 × 10	525	390	315	260	195
6 × 12	855	640	510	425	320
7 × 15	1600	1200	965	800	600
9 × 12	2000	1500	1200	1000	750

¹ From Mine Smelter and Supply Co. *Bulletin*.

For a hard ore or coarser feed than 1 in., capacities will decrease 10 to 35 per cent, and for softer ore or finer feed than 1 in., capacities will increase 10 to 60 per cent.

other crushers that enter the finer ranges of crushing down to 8 to 14 mesh. The mill is also capable of grinding damper material without packing than crusher, hammer, or ball mills. Hard and abrasive materials can be reduced with low maintenance cost. It delivers a product relatively free of tramp oversize and excessive fines.

Disadvantages.—The rod mill is not so good a fine grinding unit as a ball mill, and its range of reduction is limited. It will not handle high circulating loads satisfactorily. It must be stopped periodically to change rods or remove worn, broken rod portions to maintain efficiency. It is not so flexible as the ball mill or high-speed types of dry-grinding mills.

Cascade Mill.—The Hardinge Cascade mill (Fig. 27) is an over-all reduction device. It takes blocky run-of-mine or quarry feed and reduces it in one operation to as fine a product as wanted; *i.e.*, a maximum of 10-in. to 14-in. feed size to, say, 80 per cent minus 200 mesh. The device may be arranged for wet grinding in closed circuit with a suitable classifier. The illustration here is as a dry-grinding unit that has proved

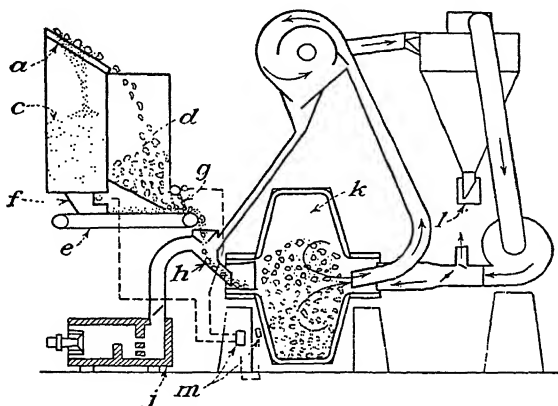


FIG. 27.—Hardinge Cascade mill.

satisfactory in grinding ores which, after grinding, are repulped for treatment by flotation or other wet-processing methods. Run-of-mine ore is delivered to an adjustable grizzly *a* which separates the feed into two main sizes *c* and *d*. Size *d* will contain only coarse sizes capable of acting as grinding media, and *c* the balance. The product of each bin is delivered to a conveyor *e* from separately controlled feeders *f* and *g* to mill feed chute *h*. If the feed is damp, hot air is also introduced with the feed from a hot-air source *j*, which on entering the mill dries the material to aid the grinding action. The drum *k* is rotated at slightly above normal ball- or pebble-mill speed to cause some free fall so as to obtain a maximum crushing and grinding effect. The method of removal of the product and classification is similar to that described for the reverse-current air-classifying system described for Fig. 33 with the final product being delivered at *l* any convenient distance above the mill that will permit of complete gravity operation in the subsequent process. Since run-of-mine or quarry sizes vary considerably as delivered, and since the ratio of large to small sizes in a mill should be kept reasonably constant if good crushing and grinding efficiency is to be maintained, an equalizing method is employed. Since grinding efficiency is a function of mill load and also sound, then by means of the "Electric-Ear" sound control, described on page 53, the feed rates are automatically regulated by the sound

of the mill so as to maintain constant and maximum reduction efficiency within the mill regardless of irregularities in the periodic introduction of coarse and fine sizes to the feed hopper. The coarse-size feeder *g* may be fed at such a rate as always to maintain some of these coarse sizes and feed them constantly as grinding media; the fine sizes are fed to the mill as called for by the "Electric-Ear" sound control *m*. The over-all effect is to secure a steady grinding rate at maximum mill efficiency commensurate with average over-all size range and character of material being ground. If an excess or deficiency in the coarse sizes occurs owing to change in ore or mining methods, then the grizzly bar spacing *a* is altered to suit, or the feed rate at *g* is changed, or both. The fine feed is then fed in at a somewhat different rate as called for by the sound of the mill. Table 16 covers data on the Hardinge Cascade mill.

Advantages.—One unit does the entire crushing and grinding operation. Over-all capital cost and floor space per ton of product obtained are low. Over-all efficiency is high particularly when considering comparative maintenance costs of equivalent crushers, conveyers, rod or ball mills, classifiers, pumps, etc.

Disadvantages.—This mill is adaptable to run-of-mine—not fine—feed sizes. It does not handle ores or materials that break down or shatter too readily such as sandstone, slate, or soft clay.

TABLE 16.—HARDINGE CASCADE MILL DATA

Mill size, diam. and length, ft.	Weight of mill and lining, lb.	R.p.m. of mill	Hp. of mill motor	Capacity, ¹ tons per 24 hr.
5 × 2	15,000	32	10	6– 10
6 × 2	20,000	29	15	11– 16
8 × 3	36,000	24	40	30– 50
10 × 4	60,000	21	100	75– 125
12 × 4	85,000	19.5	150	125– 200
14 × 5	135,000	18.0	300	250– 350
18 × 6	225,000	15.7	500	400– 650
20 × 8	325,000	15.0	900	750–1200

¹ Capacities shown are based on run of mine, minus-10-in. quartz ore and depend upon hardness and physical characteristics. The product is assumed to be 65 mesh and 70 to 75 per cent minus 200 mesh.

Feeders.—Feeders for mills are of two basic types: one removes the feed from a bin in controlled amounts, the other is used on drum mills to deliver the feed or oversize into the mill drum in any quantity as received without back spilling. Regulating feeders are of several basic designs. One type (see Fig. 34) is a revolving disk, fed at the center from an opening in the bin; as the disk revolves, the material comes out and is scraped off by an adjustable scraper into the feed chute of the mill. Another generally used type is in the form of a short belt conveyer, one end of which is under a hole in the bin. Regulation is had either by varying the belt speed, or by degree of gate opening. Both of these types are known as "volumetric feeders." To account for bulking and variation in weight, a feeder that weighs as it feeds assists in attaining maximum mill-grinding efficiency and permits the operator to secure a record of the actual tonnage fed at the same time. Such a feeder is the Hardinge constant-weight feeder illustrated in Fig. 28. The conveyer is pivoted so that as feed is moved toward the delivery point, and if over the amount that balances a counterweight, it starts to tilt down which by linkage closes down the feed gate so that less feed is delivered to the conveyer, thus compensating for the overweight, and feed rate again assumes the

rate set by the counterweight. If the weight is light, then the conveyer tends to tilt upward and the opening is increased.

Feeders on drum mills are basically simple; they are either a scoop attached to, and rotating with, the mill trunnion that dips into a stationary feed box, or a drum with a central opening for introduction of the feed, in which an internal deflector or lifter in turn delivers the feed into the trunnion (Figs. 29 and 30). The combination drum and scoop feeder is a combination of the above two types. Coarse feed is fed directly to the drum, and the oversize returning from the classifier is picked up from a point well below the mill center line, combined with the coarse feed, and delivered to the mill.

Controls.—Various control devices are playing an increasingly important role in improving grinding efficiency and the character of the product. In wet grinding in closed circuit with classifiers, density of the classifier overflow is maintained constant by one device known as the "Masco density control" which utilizes the pressure differential of an air jet in a pulp containing liquid and solids. As the density—back pressure—increases above the desired quantity, the device operates a water valve to add dilution water and bring the density back to normal.

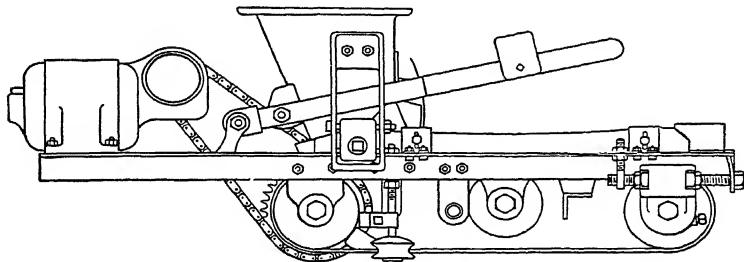


FIG. 28.—Hardinge constant-weight feeder.

In dry-grinding ball mills where the load level in the mill is an indication of mill loading, a similar arrangement has been used, but in this case the mill load is controlled by actuating the feeder.

In those cases where air classification is employed, the difference in static head in the system is an indication of classifier loading. The classifier load and mill load can be controlled by actuating the mill feeder to maintain classifier loading constant. This method is used to a large extent to control the feed of high-speed pulverizers.

Power is also used as a feeder control for high-speed pulverizers since power increases as the mill load increases and is maximum when the mill is overloaded. In the case of ball, pebble, rod, or tube mills, power as a feeder control is not satisfactory since the difference between no load and maximum grinding capacity conditions is slight, and when the mills become overloaded the power actually decreases.

The automatic control of either wet- or dry-grinding mill capacity and fineness by the sound of the mill has been accomplished by use of a device known as the "Electric Ear." As the sound of the mill increases, meaning that the mill is becoming overloaded, the "Electric Ear," through a sound-sensitive electric circuit, increases the feed rate until the sound reaches an established optimum noise level and is held at the optimum level without overloading the mill. Where a constant-weight feeder is employed, and an electrically controlled valve also is actuated simultaneously with the feeder, a wet-grinding mill circuit may be held at constant pulp density, not only as to classifier overflow, but to mill pulp density as well, regardless of changes in over-

all capacity over short or long intervals due to changes in character of feed or grinding conditions. Figure 27 illustrates the "Electric Ear" in a dry-grinding mill circuit.

Controls not only increase mill capacity and control the fineness without increase in power, but permit the use of unskilled labor or utilization of the available labor for additional duties without fear of loss of tonnage or damage to mills occasioned by negligence.

Notes on Grinding.—In addition to the necessity of reducing an ore or material to a given particle size, the shape and fineness distribution may also be important considerations. The mills described in the foregoing sections, some grinding wet, others dry, and still others either wet or dry, may be suited to limited application only. This is aside from the effect of the material on the capacity or wearing parts of the mill, which establishes grinding economy or the lack of it.

In the grinding of certain ores wet, a minimum of fines may be essential to attain maximum recovery of the mineral. This is usually true if concentrating tables are used. A mill or a method that produces a minimum of fines is then to be given first consideration. Rolls, rod mills, or short-cylinder Conical mills, all preferably operated in closed circuit with screens or classifiers, fill this requirement. On the other hand, where a highly plastic workable cement or gypsum plaster is required, then the long-cylinder dry ball-tube mill grinding in open circuit is the best device to use. For the

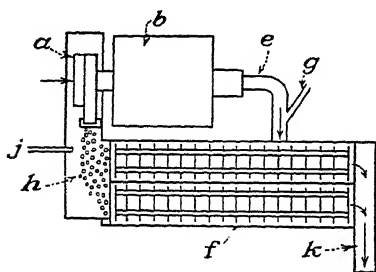


FIG. 29.—Ball mill in closed circuit with rake classifier.

usual product calling merely for a limiting maximum mesh size, then a large variety of mills are suitable, the main consideration being economy in operation to produce the product desired. Here again, whether the grinding is done wet or dry, consideration as to capital cost, power required, maintenance cost, "outage" time, and auxiliaries are all important. Of all the mills listed, the drum-type mill in its various forms is by far the most flexible and most universally used. Its use, however, becomes less attractive as the material becomes softer and is ground dry, since the high-speed pulverizer is then able to compete

with and may surpass its over-all economy. Study of the advantages and disadvantages of the various mills described will bring this point out better than further comment here.

Mill Arrangements (Layouts).—A number of mill arrangements with different types of sizing devices or methods of operation for both wet- and dry-grinding operations are illustrated.

Figure 29 is a simple plan view of a wet-grinding drum mill, ball, pebble, or rod mill, operating in circuit with a mechanical-rake classifier. The unground feed enters the central opening of the combination drum and scoop feeder at *a* and is ground in mill *b*. The product discharges through the opposite end and into launder or trough *e* where it is delivered to classifier *f* together with dilution water *g* added in an amount to allow only the oversize to drop to the bottom of the classifier where it is scraped back and above the water level and out of the classifier into inclined launder *h*. Mill water is added at *j* which, together with the oversize, is picked up by the scoop of feeder *a* and, joining the new feed, again is delivered to the grinding zone of the mill. The fine suspended solids in the classifier flow over an overflow weir into product launder *k* and out of the circuit. Fineness is then controlled within limits by the amount of water added at *g* and also at *j*. The greater the quantity of water, the

greater the dilution and tendency to drop out coarser particles; hence the finer will be the overflow unless the water volume is so high as literally to wash out all solids by its turbulence and velocity, a condition that may result if the classifier selected is too small. Water at *j* is kept at a minimum so as to cause a pulp in the mill of sufficient density to be somewhat sticky and coat the balls and lining. This increases capacity and decreases ball or rod and lining consumption.

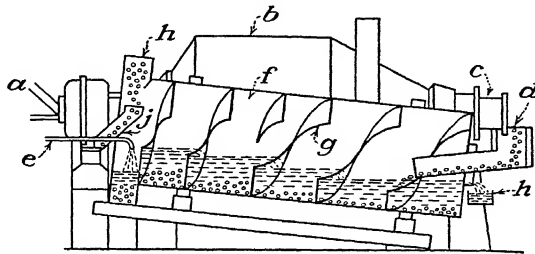


FIG. 30.—Ball mill in closed circuit with Counter Current classifier.

Figure 30 is a view in elevation of the Hardinge Counter Current classifier in closed circuit with a Conical mill. This is a shorter classifier than the one illustrated in Fig. 29, but it shows the difference in elevation between the feed and discharge end and how the mill and classifier are operated together without the need of auxiliary equipment. The feed enters at *a* and is ground in mill *b* and discharged through spout *c* into launder *d*. Classifier dilution or wash water enters at the opposite end through pipe *e* where it works counter to the settled coarse particles in the classifier drum *f*. The flights *g* attached to, and rotating with, the drum move the solids upward and into expanded chamber *h* where they are lifted by drain bucket or lifting flights and dropped into hopper *j*, which delivers the oversize into the scoop feeder box where it is again picked up and returned to the mill. The water and suspended solids pass out of the classifier at the opposite end and into the discharge launder *k*. The countercurrent washing action cleans the oversize of fine particles that may also have settled out with the result that a closely sized oversize and finished product is secured.

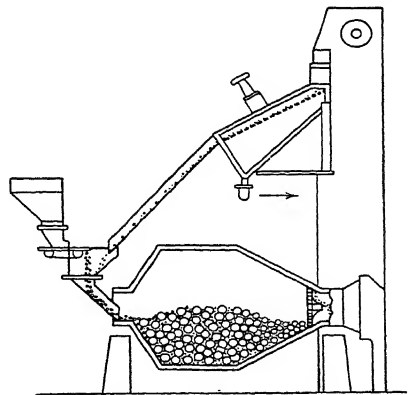


FIG. 31.—Ball mill in circuit with vibrating screen.

Figure 31 illustrates a ball mill operating in closed circuit with a vibrating screen for operation either wet or dry. An elevator taking the mill product to the screen is shown here, but a sand pump could be used for wet grinding if desired. In this arrangement, the product from the screen is at a point above the mill to permit gravity operation in the next process or delivery directly to a storage bin. The grate in the cone is relatively small and away from the direct action of the larger balls, but owing to the action caused by the cone, as described for the Conical mill, it is ample to handle

all the mill can grind, including a large circulating load as well. Screens in closed circuit with ball mills are used when grinding wet up to 10 or 20 mesh, for dry grinding up to 28 to 35 mesh, unless the case is special and high screen maintenance is secondary in importance to character of product produced which is closely sized, particularly as to particle dimension rather than weight.

A ball mill in closed circuit with a mechanical air classifier is illustrated in Fig. 32. This is an arrangement for fine dry grinding only. Products are finer than those obtained by a screen. On such materials as limestone or cement clinker, the fineness may range from about 50 per cent minus 200 up to 99.9 per cent minus 325 mesh under certain conditions. Such an arrangement as this is also used in closed circuit with long ball tube mills or compartment mills (Fig. 25) or even some of the high-speed types of mills such as the Babcock and Wilcox mill (Fig. 20).

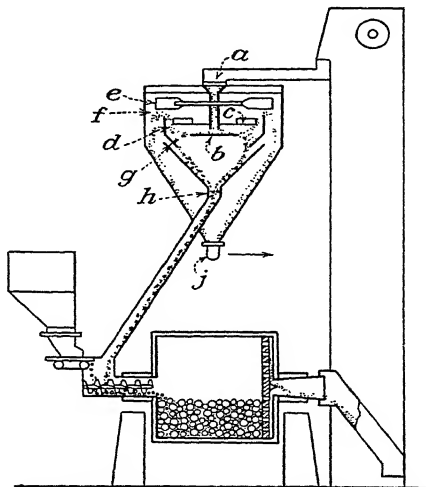


FIG. 32.—Ball mill with air classifier, elevator, and conveyor.

The mechanical air classifier is fed to *a* through a central pipe down to revolving distributing plate *b* that throws out the material radially. Also on the shaft rotating the plate are two sets of fan blades; a smaller set *c* causes a centrifugal motion of the air in the chamber *d* which throws out coarser particles that have risen with an upward moving air current produced by the large fan blades *e*. The fine material and the air that rises over the confining chamber *d* pass into the larger chamber *f* where the fine product drops out of the air stream by gravity plus centrifugal motion also occurring in this chamber as a result of the influence of the fan *e*. The air is drawn into the space back into the inner zone where it again rises and lifts the fine sizes out of the fanned-out material thrown from the feed plate *b*, thus completing the cycle. The oversize particles that fail to rise with the air stream pass down and out the central chamber at *h*. The fines are delivered from the outer casing at *j* into a conveyor or product bin. Only well-dried material can be classified satisfactorily in this arrangement. Sizes, powers, and capacities of Raymond mechanical air classifiers are given in Table 17.

TABLE 17.—RAYMOND MECHANICAL SEPARATOR (CLASSIFIER) CAPACITIES

Diam., ft.	Vertical shaft, r.p.m.	Portland cement ¹ double-whizzer type		Limestone ² single-whizzer type	
		Capacity, barrels per hr. (376 lb.)	Hp. of motor	Capacity, tons per hr.	Hp. of motor
4	400	5	10	1.5	7.5
6	350	12	15	3	10
8	275	30	25	9	15
10	250	50	40	12	25
12	225	75	50	18	30
14	200	100	75	26	50
16	185	150	100	37	75
18	165	190	125	48	100

¹ Capacity based on a feed of 85 per cent passing 200 mesh equal to 900 S.A. (900 sq. cm. of exposed surface per gram). Product 97 per cent passing 200 mesh, or about 1800 S.A.

² Limestone, or raw cement mix, capacity based on feed of 60 per cent passing 200 mesh. Product 90 per cent will pass a 200-mesh screen, or 99 per cent passing a 100-mesh screen.

NOTE: Above powers do not include those for bucket elevators and screw conveyers in separator system.

A pneumatic method of controlling the fineness in a ball or pebble mill circuit is that illustrated by Fig. 33. In this arrangement, the entire system is under a slight negative pressure to eliminate all dust, which is usually quite a nuisance in other sys-

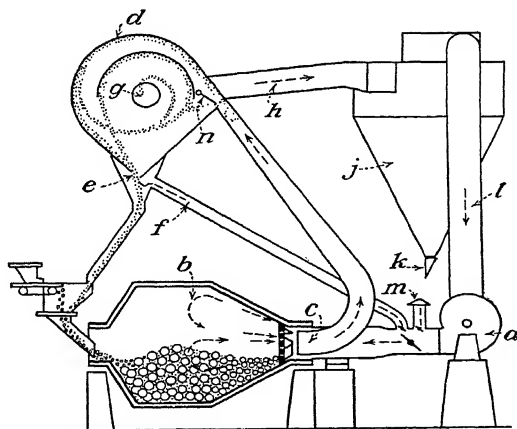


FIG. 33.—Conical mill in closed circuit swept by reversed air current.

tems not so controlled. The only moving part in the classifying system is the fan which circulates the air through the mill and classifying system and causes it to act as an elevator and conveyor, thus eliminating the necessity of using independent elevators and conveyers. The air from the fan *a* is blown into the mill where it reverses

in direction *b* and is drawn out through outlet pipe *c* into "loop" classifier *d*. Here, owing to change in direction and centrifugal force, the coarse on the outside is deflected downward and into hopper opening *e*. Here the coarse meets a countercurrent air stream from pipe *f* which cleans it of entrapped fines, carrying them back into the classifier and joining the fines that are drawn in by the air current in a cyclonic path to outlet *g* where they are conveyed through pipe *h* to product collector *j*. The separated fines are delivered by air lock *k* to any convenient elevation, 100 or more feet above the mill floor if desired. The air from the product collector is then returned to the fan by pipe *l*, thus completing the cycle except that moisture and leakage air is vented to atmosphere at *m*, or passed to a bag collector if any appreciable amount of fine dust is still present. Classification of fineness is attained by the degree of cyclonic motion in classifier *d*. This is controlled by splitter vane *n* which, if moved into the upcoming material-laden air stream, causes part of this stream to enter the classifier in the opposite direction to the main stream, slowing it down and decreasing the cyclonic effect with a resulting reduction in classifying action and a coarser product.

When damp materials are to be ground, hot air is then introduced into the mill, as indicated for the Cascade mill (Fig. 27), and vented at *m*. The system is capable of handling very damp materials; drying and grinding coal, limestone, and concentrates containing 10 to 15 per cent moisture is not unusual.

With this system, products as coarse as 14 mesh and as fine as 400 mesh can be attained depending upon air velocity, quantity, and classifier size. A combination of air classification and screened product is also obtained simply by placing a screen in

TABLE 18.—HARDINGE LOOP AIR CLASSIFIER SPECIFICATIONS

Size	Diam. of loop	Hp. to classify and elevate product 30 to 50 ft.	Capacity ¹ at fineness specified (assuming approx. 1% remaining on stated mesh)				
			20 mesh	65 mesh	100 mesh	200 mesh	325 mesh
12	12"	1	800 lb. per hr.	400 lb. per hr.	300 lb. per hr.	250 lb. per hr.	175 lb. per hr.
24	24"	3	2800 lb. per hr.	1400 lb. per hr.	1000 lb. per hr.	800 lb. per hr.	600 lb. per hr.
36	3'	7½	3.5 tons per hr.	1.7 tons per hr.	1.3 tons per hr.	1.0 tons per hr.	0.7 tons per hr.
54	4½'	15	7 tons per hr.	3.4 tons per hr.	2.0 tons per hr.	1.6 tons per hr.	1.0 tons per hr.
72	6'	30	14 tons per hr.	7 tons per hr.	4.0 tons per hr.	3.0 tons per hr.	2 tons per hr.
108	9'	60	28 tons per hr.	14 tons per hr.	10 tons per hr.	8.5 tons per hr.	6 tons per hr.
126	10½'	100	48 tons per hr.	24 tons per hr.	18 tons per hr.	14 tons per hr.	10 tons per hr.
144	12'	125	60 tons per hr.	30 tons per hr.	22 tons per hr.	18 tons per hr.	12 tons per hr.
168	14'	200	90 tons per hr.	45 tons per hr.	34 tons per hr.	27 tons per hr.	20 tons per hr.

¹ Capacities are subject to variation due to temperature, moisture, elevation, nature of material, and local conditions.

the oversize return circuit from the classifier; thus two or more closely sized products can be obtained simultaneously. Air sweeping the mill increases mill efficiency and capacity over the trunnion-overflow discharge arrangement. A better control over the character of product is had as well. Table 18 covers data on the Hardinge loop air classifier.

Figure 34 illustrates a typical unit pulverizer where coal is ground, then blown into the furnace. The arrangement shown serves to illustrate the basic principle of operation as applied to high-speed mills, as well as unit ball-mill pulverizers.

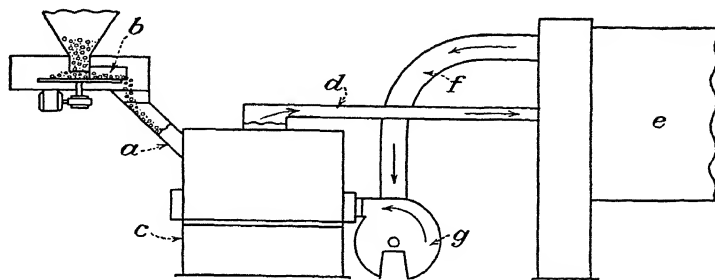


FIG. 34.—Unit coal pulverizer for direct firing.

Feed enters at *a* from regulating enclosed feeder *b*. The coal, on being pulverized in the mill *c*, is air swept out of the grinding zone into a classifier that eliminates the oversizes and carries the fines through pipe *d* to a burner in the furnace, boiler, or kiln *e*. Hot air is drawn from the hot walls, pit, or flue through pipe *f* to fan *g* which causes it to pass through the mill and dry the coal in the mill while being ground, thus eliminating the necessity of a separate dryer.

Unit pulverizers of the high-speed type are used when grinding soft bituminous coals. Conical ball mills are used when grinding the harder grades of bituminous coals, or anthracite, and also when long, uninterrupted runs are essential without fear of unscheduled shutdowns as is required in firing reverberatory furnaces. Capacities are substantially the same as those specified in various tables.

CHAPTER IV

SAMPLING

BY DONALD M. LIDDELL¹

Definition.—Sampling is the process of obtaining a small quantity of material that shall be representative of the average of some large amount. Theoretically, each portion of a sample, however small, should itself be representative of the large body, but in practice this condition cannot be obtained, owing to the impossibility of reducing all the sample to a molecular state of subdivision.

Owing to the difficulties of securing a small sample, weighing only a few ounces, which will represent a large amount of material, the sampler will ordinarily divide a large shipment into several lots, each to be sampled and settled for separately.

The importance of proper sampling is rarely correctly estimated. The utmost care is ordinarily given to all the operations of a chemical laboratory. Large sums are spent in its proper equipment, and high salaries are paid to the chemists employed, without regard to the fact that the accuracy of the analytical work is primarily limited by the accuracy of the samples furnished to the laboratories. It appears to be the customary assumption that any \$3-a-day man can prepare samples for a \$20,000 staff of chemists. As an interesting commentary on how little is ordinarily thought of sampling, it was not until the eleventh volume of the *Transactions* of the A.I.M.E. that any article appeared on sampling.

The accuracy of a sample can be vitiated in many ways, but a proper understanding of the underlying principles will enable one to see the mistakes and to correct them. On the other hand, one of the easiest ways to point out the underlying principles is to describe certain specific errors in sampling and to show the cause and the cure, so that in large measure this chapter will be a discussion of the errors of sampling.

In the sampling of solid materials, two classes at once are apparent—those which can be crushed, and those which must be drilled, sawed, or filed. In sampling materials of the first class, where payment must be made on the value of the shipment, the question of moisture is the first to present itself and is probably one of the greatest sources of error. After weighing the material, the moisture sample must be taken as quickly as possible, and it must be large enough to represent the whole mass correctly, yet small enough to permit of drying the entire moisture sample. The reason for this stipulation is that it is impossible to crush and mix a moisture sample, as mechanical work performed on it alters the moisture content. The moisture sample must not be representative of fine material alone or of the coarse material only, but must be fairly representative of both classes in their proper proportions. Usually, a finely divided material will carry a greater proportion of moisture than will coarse lumps, and therefore the sampler, because of the small size of the moisture sample (comparatively), usually takes an undue proportion of fines, since even one lump of ore appears as a mountain in the moisture sample.

Ordinarily, a moisture sample is said to have been dried at 212°F., even though the temperature of the drying oven may have been allowed to run even to 240°. The drying temperature is important, for the material may contain water of crystallization

¹ Consulting engineer, New York, N.Y.

not completely given off at 212°, or it may contain salts in which the acid radical is more or less completely expelled at temperatures only slightly above 212°, or the "moisture" may be some such substance as free sulphuric acid. At any rate, it is important that the moisture sample and the final sample for the laboratory should be dried at the same temperature, and if the material is hygroscopic it must be kept in a stoppered bottle or other watertight receptacle. It should also be realized that practically all crushed pulps are hygroscopic, and if any sample is to be reanalyzed after it has stood for some time in paper or cloth bags, it should be dried first.

In drawing the original moisture sample from any material whatever it is obvious that the sample must not be taken from the immediate surface of the pile, since there will be some alteration in the moisture content after any exposure, however short. It can be proved fairly conclusively that, if the moisture sample is taken after the first crushing on material the size of which is fairly uniform, the moisture thus determined will be one-tenth to one-fourteenth lower than it should be, depending on the season of the year. Thus in ordinary summer weather, if a value of 5 per cent is obtained for the moisture, the true moisture is probably close to 5.5 per cent.

In taking a sample for laboratory purposes, rarely is the entire material either put through a sampling mill or hand sampled. A certain proportion is ordinarily withdrawn at once and put through the sampling process. This is primarily a matter of sampling cost, but may have important commercial considerations also. For instance, in sampling ore for smelting it is usually desirable to keep it as coarse as possible. On the other hand, ore to be milled can be crushed *in toto* previous to sampling. To determine this point, the fineness of crushing, the sampler should know what is to be done with the material, but it is preferable to take the selection of the material that enters the sample away from the judgment of the sampler, so that machine sampling is best. It is at least imperative that a sampler take every fifth or tenth or twentieth shovelful rather than a shovelful whenever his inclinations so move him, though Brunton says that at best the process is that of "shovel selection" rather than "shovel sampling." Also the shovelfuls should be about the same size—both of the material accepted for the sample and of that going into the reject—and here the working period comes in, for at the close of a long period of shoveling, the shovelfuls will usually be much smaller than at the start.

Other plans than taking shovelfuls at stated intervals are necessary where material is not being unloaded by hand, but in any case the principle remains that the selection of the sample should not be left to the caprice of the sampler—thus the U.S. Steel Corp. in sampling ore arriving in hopper-bottom cars specifies that a network of cords shall be stretched over the top of the car and that equal quantities of material shall be taken from under each of the points of intersection in the net.¹ In sampling certain homogeneous manganese ore arriving by steamer, it is perfectly proper to take every twentieth or even fortieth scoop as it comes up through the hatch. The sample scoops are then unloaded into a car by themselves, and that car alone later goes to a sampling mill.

It is also possible to take samples direct from the faces of ore in a vessel that is being unloaded, but this method is not ordinarily to be recommended, as it leaves too much to the judgment of the individual samplers.

Of course, these fractional-selection methods introduce hand sampling, and the more that can be eliminated, the better the results. While the advocates of hand sampling claim that with a trained gang of men the actions are practically mechanical, this is untrue, for if they are trained they know what they are doing and why they are doing it, which virtually is allowing the ore to be handled by interested parties. In

¹ The full paper on the methods of the U.S. Steel Corp. for sampling ore in bulk was published in the *Jour. Ind. Eng. Chem.*, 1909, p. 107, and should be referred to by anyone interested in this subject.

such cases, it is either impossible for any man to submerge completely what he considers are his employer's interests, or else he consciously stands so straight as to lean over backward. The greatest errors introduced by hand sampling are: (1) In fractional shoveling, the shovelful is ordinarily not uniform, the ordinary tendency being to make the sample shovelful the smallest. The size of the shovelfuls will also vary through the day, as already mentioned. Moreover the place from which the sample is taken is at the discretion of the workman. (2) In coning, the larger pieces of ore invariably fall to the outside of the pile, while the top retains the fines. Then when the pile is spread out, the fines go to the very outside. As the moisture sample is ordinarily taken from near the surface of the pile and as at this stage of the work the fines contain more moisture, this operates in favor of the buyer. (The process of "coning" is explained below.)

Cutting Down Samples.—The sizes of the individual pieces and the amount of the sample itself are next alternately reduced. Almost never should there be two successive reductions in the amount of the sample without an intermediate reduc-

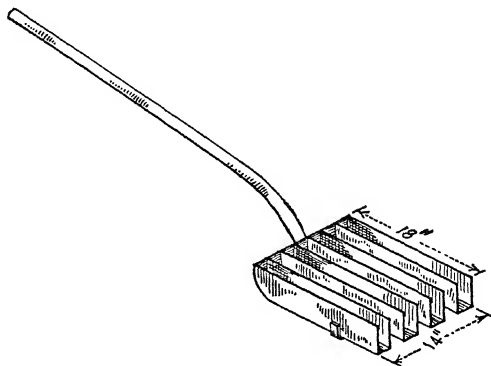


FIG. 1.—Assayer's split shovel.

tion in the size of the largest pieces making it up. It is not always necessary to crush the entire sample. Much time and expense can often be saved by screening a sample and crushing only the coarse portions, mixing them when crushed with the finer pieces that have by-passed the crushers. (The question of coarse pieces that will not crush will be discussed later under the heading, *Metallics*, page 85.) It is self-evident that after this process pains must be taken to mix the portions thoroughly.

The bulk of the sample itself may be reduced either mechanically or by hand, or by a combination of the two processes. The hand operation is ordinarily known as "coning and quartering." The material is piled first into a cone and then the top of the cone raked out so as to form the frustum of a cone.¹ Diameters are drawn across this surface at right angles, and the opposite quarters are rejected. The remaining quarters are then again shoveled into a cone, shovelfuls being taken alternately from the two quarters. This new cone is again flattened out and the quartering process repeated.

The two great dangers in this process are: Large lumps tend to segregate at the bottom and on the outside of the cone, and unless great care is taken to place the

¹ A detailed description of this process is given by Scobey, *Met. Chem. Eng.*, February, 1913.

shovelfuls exactly on the apex of the heap, there will be an undue concentration of coarse material in some one quarter of the pile. Also in quartering, the workmen will either invariably take or reject all the large lumps that lie along the border line of the cut, thus causing the sample to contain an undue amount of either coarse or fine. In raking out the cone into a frustum the radial distribution of the material should not be disturbed. This question will be referred to again under Salting (page 86) in discussing the operation known as "drawing the center." Dust should not be lost from

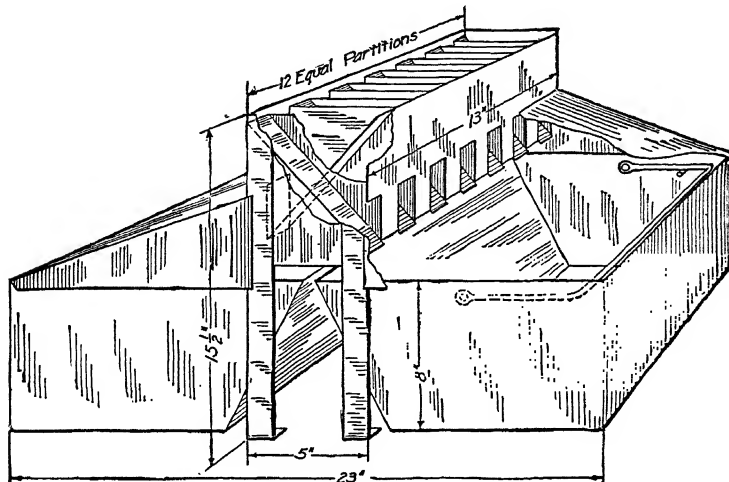


FIG. 2.—Jones sampler.

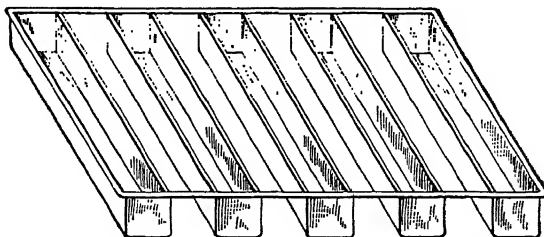


FIG. 3.—Assayer's riffle.

the material, and it may be necessary to sprinkle the pile from time to time in order to avoid this.

The assayer's shovel, shown in Fig. 1, is used in cutting down samples by hand work. This apparatus consists of a number of parallel troughs attached to a shovel handle, the spaces between the troughs equaling their widths. When crushed ore is shoveled onto this shovel, a portion is retained in the trough and a portion falls through, thereby splitting the sample.

A variation of this idea is found in the Jones sampler, which consists of a number of parallel triangular troughs facing in alternate directions, shown in Fig. 2. One-

half of the material must go one way and one-half the other. For cutting down very small and finely ground samples, still another modification is the assayer's riffle shown in Fig. 3.

The mechanical samplers ordinarily found are of the Vezin, the Bridgman, or the Brunton type,¹ or some modification of them. In any case the sampling principle is to cut across a falling stream of ore, taking either successive portions of the stream as the sampler cuts across it or else diverting the entire falling stream for a certain period into the sample. In either case the danger is the same—that the apparatus used should not be sufficiently large for the component parts of the ore stream; *i.e.*, the large lumps of ore are driven into the large rejected portion, leaving an undue proportion of fines in the sample. Also the entire falling stream should be diverted or cut across and not merely a portion of it, for such a stream tends to classify itself as it falls, and a sample cut from one edge only will not be the same as one cut from the center, nor will the edges themselves ordinarily be alike. In rotating samplers it is, therefore, important that the cutting edges are radial planes.

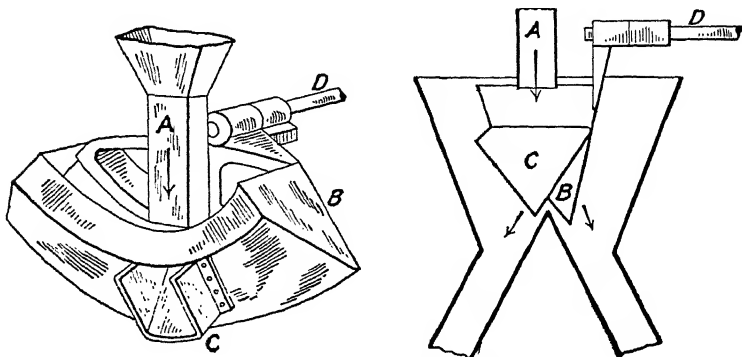


Fig. 4.—Brunton ore sampler.

The matter of relative size of piece and size of opening is illustrated very well by the sampler's shovel and the assayer's riffle. It is easy to see that these pieces of apparatus are entirely unreliable on material containing pieces of larger diameter than the widths of the openings, for then these larger pieces will all be retained on the shovel or riffle. It does not seem safe to use these two pieces of apparatus with material the maximum pieces of which exceed one-fifth the width of the opening, and it seems probable that the same proportion would be a good one to apply to sizes of the lumps and cut-out holes in machine sampling.

The Brunton sampler consists either of a wedge-shaped deflector plate that can be thrown back and forth in such manner as to send all the ore stream to a storage bin or to the sample, or of a scythe-shaped casting that swings back and forth in the ore stream. The drive is ordinarily by a connecting rod driven from a belted wheel, and by altering the position of the connecting rod the arc of travel is altered.

In the latter form most of the surface slants toward the storage-bin hopper, the remainder toward the sample bin, and the stream of ore is directed according to which portion of the casting it is hitting. The proportion of ore deflected is altered by adjusting the arc of travel of the casting. As the arc is cut down, the portion inclined

¹ The original Brunton apparatus is described in *Trans. A.I.M.E.*, Vol. 13, p. 639.

toward the sample bin constitutes a greater and greater portion of the part of the casting in use. In the former type of apparatus, the wedge, the adjustment is made by cams, which may be set at varying angles. The sampler points one way during one part of the revolution and in the other during the remainder, so that any adjustment is possible.

Practically the same result as is attained by the oscillation of the Brunton sampler is gained by rotating in the Snyder sampler (this is not the Charles Snyder sampler spoken of below). The Snyder sampler consists merely of a circular casting shaped much like a miner's gold pan, having one or more openings (according to the percentage of sample desired) in its sloping flange, mounted on the end of a horizontal shaft provided with tight and loose driving pulleys.

In operation the sampler revolves ten to thirty times per minute, the speed depending upon the size of the sampler itself and the size of the material being handled. The material to be sampled is directed by a spout so as to fall inside of the sloping flange of

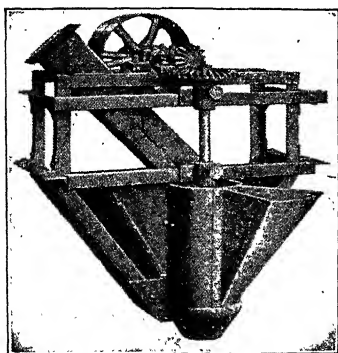


FIG. 5.—Vezin ore sampler.

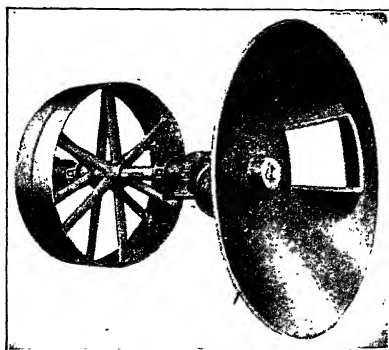


FIG. 6.—Snyder ore sampler.

the sampler. The rejections slide off the flange into a receiving hopper. The sample drops through the opening in the flange as the opening passes under the spout. It is a most simple machine, and its only disadvantage is that the percentage of sample to reject is fixed by the ratio of the arc of the opening to the full circumference.

The Vezin sampler is shown in Fig. 5, and, as will be seen, consists of a rotating cylinder bearing two or more scoops, which, as they revolve, cut through a falling stream of ore, delivering the portion they cut out (the sample) through the truncated cone at the bottom, while the reject passes on the ore bin. Theoretically, this sampler, as well as the Charles Snyder¹ and the Bridgman² types, has the objection that centrifugal action disturbs the selection. In addition, the edges of the cutting scoop occasionally are not radial, resulting in improper selection. Also, unless they are rugged in construction the edges of the scoops become bent, leading to improper results. These machines are often made so that several reductions may be made in the bulk of the sample without any intermediate crushing. This, the writer believes, is absolutely wrong. On the other hand, these samplers have great capacity, require little head room, and are easily cleaned with a hose, if necessary.

Crushing for sampling may be done in any apparatus that fulfills the following conditions: It must be easy to clean; it must not wear greatly, otherwise the material

¹ *Trans. A.I.M.E.*, Vol. 40, p. 570.

² *Trans. A.I.M.E.*, Vol. 20, p. 416.

from the crusher will contaminate the sample; the material must not fly about, but must all be delivered at one spot. Hence, gyratory crushers, jaw crushers, and rolls are used for sample breaking, and ball mills, drug mills, small breakers, and small rolls for final crushing. These three conditions are the only essentials.

For feeding material to the crushing or to the conveying machinery, it should be remembered that shaking belts or tables all classify the material fed and hence tend to inaccuracy unless pains are taken to offset this tendency.

Coal Sampling.—Probably coal sampling is subject to greater proportional errors than is any other common sampling operation, not only because coal is a cheap substance, which means that not much can be spent upon sampling it, but also because of its deceptive appearance. The various parts of a heap of coal look very homogeneous, and it usually comes already partly crushed, but the fact remains that the assumption of uniformity leads to bad errors. One swallow will come nearer making a summer than will one lump of coal, picked out by the master mechanic, to making a good sample.

While the final preparation is a matter of ordinary coning, quartering, and breaking down, the following, by C. E. Scott, is a good description of what ought to be done in taking the sample:

"The size of the sample shall not be less than the proportion of one part per 1000, and the minimum size of any one sample shall be 100 lb.; the maximum number of tons one sample shall represent shall be 1000, and preferably much smaller than this, say, 250 tons. All the impurities, such as slate and pyrites, should be broken down in the original sample, and if this is done as each shovelful is taken it will make the work much easier. In collecting the original sample from a railroad car it should be taken as the coal is being discharged; where the coal is delivered to the point of sampling in wagons, one large shovelful should be taken from each wagon; where it is delivered in a barge, the sample should be taken as the barge is discharging (if the discharging is accomplished by the aid of a bucket or belt conveyor it is advisable to take advantage of this and obtain a small shovelful from the buckets or off the belt at regular intervals); where the sampling must be taken from a pile, a shovelful should be taken from all accessible parts of the pile, and the shovel should be filled from the coal underneath the surface. As the coal is collected it must be placed in a box, barrel, or can, of sufficient size, which has been thoroughly cleaned out before using. A lid should be kept on the container at all times to prevent dust, ashes, etc., from collecting on the sample. When the total sample is completed it should be put through a crusher or dumped on to a clean wood or cement floor, or on a canvas cloth, and crushed down by means of a hammer until no pieces larger than 1 in. remain. The sample should then be mixed by shoveling it over itself into a conical pile."

Size-weight Ratio.—Many attempts have been made to deduce a proper ratio between the size of the largest pieces in the lot of material to be sampled and the size of the sample. Probably one of the best of these was a paper on Accuracy in Sampling Coal.¹ This was an investigation into the cause of inaccuracies in sampling, based on both laboratory experiment and theoretical calculation. While the investigation was concerned with coal, the general results and reasoning are applicable to all substances, as shown in the Tables on page 67.

In order to obtain a sample that shall correctly represent any material, it is necessary to crush to such a fineness that a particle more or less of any component of the material taken into the sample shall not materially affect its analysis or its value. The maximum error is determined by the ratio of the weight of the largest particle of metal or high-grade mineral or impurity to the weight of the entire lot. The chance of finding a full-size piece of highest grade material would be much greater in a lot of ore

¹ PAILEY, *Jour. Ind. Eng. Chem.*, 1909, p. 161.

crushed to 0.25-in. cubes than in a lot crushed to 1-in. cubes. Therefore, accuracy demands that the ratio between the weight of the largest particle and the entire lot shall increase directly as the fineness.

SIZE OF SLATE CONTAINED IN COAL, AND SIZE OF ORIGINAL SAMPLE REQUIRED TO ENSURE THE ERROR OF SAMPLING BEING LESS THAN 1 PER CENT

Size of slate, inches	Weight of largest piece of slate, pounds	Original sample should weigh, pounds
4	6.7	39,000
3	2.5	12,500
2	0.75	3,800
1½	0.38	1,900
1¼	0.24	1,200
1	0.12	600
¾	0.046	230
½	0.018	90

SIZE TO WHICH SLATE AND COAL SHOULD BE BROKEN BEFORE QUARTERING SAMPLES OF VARIOUS WEIGHTS

Weight of sample to be divided, pounds	Should be broken to, inches	Weight of sample to be divided, pounds	Should be broken to, inches
7,500	2	40	2 mesh
3,800	1½	5	4 mesh
1,200	1	½	8 mesh
460	¾	¼	10 mesh
180	½		

LIMIT BEYOND WHICH SAMPLES SHOULD NOT BE DIVIDED WHEN CRUSHED TO DIFFERENT SIZES IN LABORATORY

Size of coal mesh	Should not be divided to less than, grams
2	8,300
4	1,100
8	120
10	55
20	3
	Should be pulverized to at least 60 mesh

In gold-ore sampling, therefore, it is customary to go a little beyond the above figures, both in size of sample and in fineness of crushing, while in sampling chemicals and other more uniform products the samples may be smaller in weight. Tables for ore sampling follow:

WEIGHTS TO BE TAKEN IN SAMPLING ORE¹

Weights		Diameters of largest particle					
Grams	Pounds	Very low grade of uniform ores, milli-meters	Low grade ores, milli-meters	Medium ores		Rich ores, milli-meters	Rich and spotty ores, milli-meters
				Milli-meters	Milli-meters		
	20,000.0	207.0	114.0	70.2	50.8	31.6	5.4
	10,000.0	147.0	80.3	53.9	35.9	22.4	3.8
	5,000.0	104.0	56.8	38.1	25.4	15.8	2.7
	2,000.0	65.6	35.9	24.1	16.1	10.0	1.7
	1,000.0	46.4	25.4	17.0	11.4	7.1	1.2
	500.0	32.8	18.0	12.0	8.0	5.0	0.85
	200.0	20.7	11.4	7.6	5.1	3.2	0.54
	100.0	14.7	8.0	5.4	3.6	2.2	0.38
	50.0	10.4	5.7	3.8	2.5	1.6	0.27
	20.0	6.6	3.6	2.4	1.6	1.0	0.17
	10.0	4.6	2.5	1.7	1.1	0.71	0.12
	5.0	3.3	1.8	1.2	0.80	0.50	
	2.0	2.1	1.1	0.76	0.51	0.32	
	1.0	1.5	0.80	0.54	0.36	0.22	
	0.5	1.0	0.57	0.38	0.25	0.16	
90.0	0.2	0.66	0.36	0.24	0.16	0.10	
45.0	0.1	0.46	0.25	0.17	0.11		
22.5	0.05	0.33	0.18	0.12			
9.0	0.02	0.21	0.11				
4.5	0.01	0.15					
2.25	0.005	0.10					

¹ RICHARDS, R. H., "Ore Dressing," 2.

Many experiments on the sampling of gold ores lead to the belief that carrying the grinding beyond 100 mesh does not add to the accuracy of the sampling and that at this fineness equally good results will be obtained whether the material to be taken for

SIZE-WEIGHT RATIO IN SAMPLING¹

Diameter of largest particle, inches	Minimum weight of sample, pounds, Colorado practice
0.04	0.0625
0.08	0.50
0.16	4.00
0.32	32.00
0.64	256.00
1.25	2,048.00
2.50	16,384.00

¹ SMITH, E. A., "Sampling and Assay of the Precious Metals.

the assay is cut out of the sample by one cut of the spatula or taken by various cuts from over the entire surface of the sample.¹

SMALLEST PERMISSIBLE WEIGHT FOR SAMPLES OF A GIVEN SIZE¹

Size, inches cube or mesh	Weight of sample, pounds	Ratio of weight of largest cube to weight of sample	Effect on value created by one cube assaying \$100,000 per ton, of sp. gr. 5
2	10,000	1: 7,000	\$14.42
1½	5,000	1: 8,300	12.17
1	2,000	1: 11,000	9.00
¾	1,000	1: 13,000	7.50
½	400	1: 18,000	5.62
⅜	300	1: 31,000	3.17
¼	200	1: 71,000	1.40
⅜	100	1: 83,000	1.20
⅛	75	1: 220,000	0.44
6 mesh	50	1: 430,000	0.23
10 mesh	25	1: 930,000	0.107
18 mesh	10	1:1,900,000	0.051
30 mesh	4	1:4,200,000	0.023
50 mesh	1	1:5,500,000	0.018

¹ RICHARDS, R. H., "Ore Dressing," Vol. 3.

SCHEME FOR SAMPLING RICH ORES WITH VEZIN SAMPLERS¹

	Inches	Sample, per cent	Pounds in 100 tons
Maximum size of cubes.....	1 00	20.0000	40,000
Maximum size of cubes.....	0.25	1.2500	2,500
8 mesh.....	0.0625	0.0785	157
30 mesh.....	0.0171	0.0050	10

¹ RICHARDS, R. H., "Ore Dressing," Vol. 3.

Therefore it would seem that 100 mesh is as fine as it is necessary to reduce any samples, so far as accuracy of sampling is concerned. On the other hand, since more concordant results have been obtained on a 60-mesh sample after reducing it to 80 mesh, it may be concluded that 80 and 100 mesh mark the limits of desirable sampling. This totally ignores the question as to what are the best limits for manipulation from the chemist's and assayer's side. It may safely be assumed that no material requires finer grinding than does a spotty gold ore.

It is also of importance to note that contamination of the sample with material abraded from the grinding machinery increases rapidly as the mesh of the sample is cut down, so that this in itself offers a reason for keeping the mesh size as large as

¹ WRIGHT, *Mining Mag.*, November, 1910.

The contamination of samples, even of materials that can be easily crushed, by particles of the sampling tools may reach greater proportions than is ordinarily credited. In sampling coke using a Braun pulverizer, the writer has shown that the amount of iron introduced in the final sample may equal 1 per cent of its entire weight, all of which will, of course, show as ash when that constituent is determined, meaning that this determination will be at least 6 per cent from the true result and may be as high as 20 per cent. From the ordinary bucking board and muller used for ore crushing by a not overindustrious Mexican, iron to the extent of 3 per cent of the weight of the sample was introduced into it. Of course, if the sample is to be assayed for copper, lead, silver, or gold, the introduction of iron in this way saves a considerable amount of money in the course of a year in payments for these valuable metals, but the inaccuracies introduced are fatal to good furnace practice, even if one has chloroformed his conscience.¹ For this reason the selection of material for the grinding tools to be used in sampling is a most important question. Chrome steel and manganese steel should replace the cast iron that is usually used, and, incidentally, the greater wear obtained from these harder substances will almost more than pay for the difference in original prices. For this reason, too, bucking boards and mullers that have been cast with chilled surfaces should be watched, and when the chilled surface is worn through they should be promptly discarded.

That iron can be introduced to the extent of 1 per cent of the weight of the coke sample may possibly be questioned. There is no doubt, however, that in the samples spoken of this amount of iron was present in a metallic state as was determined by agitating the sample with a neutral copper sulphate solution, washing the resulting mass free from the resultant copper and iron salts, and determining the metallic copper left precipitated on the coke dust. The iron equivalent of this amount of copper was then calculated. The metallic iron thus shown to be present was not iron reduced from the ashy constituents of the coke by the action of the incandescent carbon during the coking process; for when the same coke was ground in Abbé mills or crushed in porcelain and agate mortars using porcelain and agate pestles, the sample thus produced did not react with neutral copper sulphate solution, nor was the analysis of the ash obtained by burning off the coke at all the same, as the latter case showed much more silica and much less iron. Reference to this subject of sample contamination will be made again under copper-bullion sampling.

SAMPLING METALLIC MATERIAL

In sampling metals there are two main dangers to be guarded against: The outside of the bar or slab will ordinarily have a certain amount of dirt adhering to it of a character entirely dissimilar from the inside of the bar, and also in the interior of the bar the metal is as a rule not homogeneous. It is the rare exception when an alloy of two or more metals does not segregate upon cooling.

In fact, it is only eutectics that will not segregate on cooling, so that usually when an alloy cools down some definite compound tends to separate from it and solidify before the remainder freezes. Thus, in an alloy of pure copper and silver containing less than 0.5 per cent Ag, the copper tends to crystallize out on slow cooling, leaving an enriched silver-copper alloy behind. With 0.5 to 1.0 per cent Ag the alloy solidifies fairly homogeneously, but with over 1.0 per cent Ag the silver shows a tendency to crystallize out first, leaving a poorer alloy behind. (There is a true eutectic at 71.89 per cent Cu, 28.11 per cent Ag, but such bullions are negligible in commerce.) This segregation may, however, be reversed by the addition of impurities.

¹ The introduction of metallic iron into the sample also results in the assayer's finding a heavy black residue when the sample is panned that not infrequently results in everyone starting to look for iridium. It also prohibits the use of evolution methods for the determination of CO₂ in the ore.

An illustration of such reversal is given by Dr. Edward Keller in *Bureau of Mines Bulletin* 122, where he gives figures showing the distribution of silver in gas-finished converter copper and in black-copper bars, the homogeneity of the latter being unintentionally brought about by the latter being almost saturated with cuprous sulphide and other impurities. It follows that, in producing converter copper, the bars can be made much more homogeneous and much sampling difficulty avoided by either underblowing the copper (leaving it saturated with Cu_2S) or overblowing (intentionally saturating with Cu_2O). Figures 8 and 9, from Dr. Keller's work, show this graphically.

As regards dirt and scale on the outside of the bar, there are times that a portion of this belongs in the sample, as in sampling copper or lead bullion, where the metal is to be paid for upon assay—as the dirt is weighed with the shipment, a certain portion of it should appear in the sample. On the other hand, in handling bars of bullion of any sort, it is very easy to detach more of this dirt or scale, and get it into the sample pan, than should properly be there. Here, then, is a case in which the judgment of the representatives of buyer and seller must be used to determine when the proper amount of dirt is getting into the sample and when this is being exceeded.

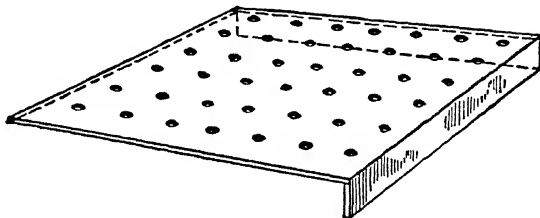


Fig. 7.—Template for drilling sample holes.

The dirt and scale on the outside of metallic bars give rise to some very curious consequences. It would appear at first sight that in the drilling of copper bullion it would make no difference whether the drill was pushed through the bar from top to bottom or vice versa, yet this is not the case, nor is it by any means merely a theoretical difference. The gold and silver contents may be affected by more than a per cent of their value and the copper content itself sensibly modified by the direction in which the drilling is done. This is due not alone to the fact that ordinarily the top of the copper bar will have more dirt and scale on it than will the lower surface, but also to the fact that the interstices in the metal will retain some of the fine powder first drilled out when the metal is being drilled top up, while all this powder and the dirt will fall into the sample when the bar is being drilled bottom up.¹

The fine powder is not a negligible consideration. Apart from the zone of enrichment where a bar of copper has last solidified (about one-third of the distance from the top to the bottom), the top skin of a converter bar is always much richer than the metal below it. Dr. Keller (*loc. cit.*) believes that toward the last of the blow some of the silver compounds become oxidized and insoluble in the metal and rise to the top of the bar. P. O. Wels believes it to be a spray carried to the surface by the gas from the parts that freeze last. At any rate, if dirt and oxide are collected from the top surface of converter bars, they will be found often to carry more silver than the drilled sample itself, so that occasionally the seller is benefited when an undue amount of surface dirt enters a copper sample.

¹ *Eng. Mining Jour.*, Nov. 5, 1910, p. 897.

Consequently, bullion sampling becomes a continuous struggle to obtain correct results in spite of accidental contamination from dirt, intentional contamination by dishonest representatives, and the effects of segregation.

It should be unnecessary to point out that, when wooden blocks are used under material that is being drilled, care should be taken not to drill into these blocks, but too many samples are found containing wood chips to allow this caution to be omitted.

In the sampling of lead bars a punch is ordinarily used, so that this question of dirt does not enter in any great degree. On the other hand, the punch is ordinarily driven halfway through the bar, so that it is important in order to avoid the effect of segregation that the available length of the punch should not be greater than half the thickness of the bar, for if it is, it results in an undue amount being taken representative of the center of the bar as compared with that representative of the outside. In punching lead bars, the sampler sometimes wets the punches with oil. As the punchings are later melted down and small sample bars cast from them, this practice results in inaccuracies, for the oil will reduce the oxide of lead present and give a higher assay than should be obtained. On the other hand, if the samples are allowed to remain uncovered too long before melting, the oil causes any flying dirt to stick to the punch cores and the sample will be contaminated in this way.

The melting is ordinarily done in a graphite crucible. Of course, this in itself tends to a certain reduction of lead oxide, but probably not more than is sufficient to overcome the effect of the oxygen of the air. This latter reagent makes it necessary that this melting should be done as quickly as possible and at the lowest temperature at which the molten lead will be completely fluid, for if the temperature is too high or the mass is kept too long in molten state the lead will scorch, resulting in the sample being higher in gold and silver than the bars actually are.

Samples of gold and silver are ordinarily poured from the molten metal, and consequently there is but little danger of contamination by particles of the sampling tools. Lead is so soft that it also is not affected. With copper, iron, steel, and some other metals the sample is invariably contaminated with particles from the sampling tools. Thus, in copper drillings that have been ground between cast-iron grinding surfaces, the contamination may amount to as much as 0.4 per cent. It might appear that in this case it would be comparatively easy to pass the drillings over a magnet and remove all this extraneous material. Unfortunately, if this were done, an error would be introduced on the other side, because blister copper will have a certain amount of converter slag adhering to it, some of which will enter the sample, where it properly belongs.¹ This slag is, like iron, paramagnetic and will be removed by a magnet. In sampling steel or iron the case is even worse. The only accurate method is to use sampling tools of a hardness greatly in excess of that of the metal to be sampled and, if possible, to ascertain the loss experienced by the tools as compared with the total weight of the sample, using these weights and the analysis of the tools and the sample as the basis for the correction. Of course, such refinement as this is happily seldom necessary.

As to the error that complete removal of magnetic material would introduce in copper sampling, the table on page 73 shows the amount and the assay of some magnetic material properly belonging to the material.

On long tests, using taper drills and chrome-steel grinding surfaces in Hance Bros & White drug mills, the conclusion was reached that about 0.012 per cent of iron was introduced into copper samples.

However, Keller, at Raritan, found the contamination to be only 0.0059 per cent, and the American Smelting & Refining Co. found it to be 0.0050 per cent. The writer made the suggestion at one time that, since practically all this came from the

¹ *Eng. Mining Jour.*, Oct. 15, 1910, p. 752.

MAGNETIC PARTICLES IN PIG-COPPER DRILLING

Weight of sample, grams	Weight of magnetic particles, grams	Loss in weight by drills, grams	Bullion assay			Assay, magnetic particles		
			Cu, per cent	Ag, ounces	Au, ounces	Cu, per cent	Ag, ounces	Au, ounces
26,470	3.805	0.56	99.05	75.65	14.035	46.95	47.3	4.30
143,390	12.050	0.94	97.825	352.2	8.375	34.00	143.8	13.60

cones used in the final grinding of the drillings, all magnetic particles be removed after the coarse grinding of the drillings, and again after the fine, and that all the latter particles should be discarded and the former (containing the converter slag) should be again mixed in. Keller believed that in bullions carrying iron, nickel, and arsenic some magnetic compounds might be thus taken out, which should remain in the sample.

It will occasionally happen that metallic material will be so dirty from adherent slag, matte, or other foreign material that it is necessary to melt the entire shipment down, recast it, and sample both metal and scoria separately (see paragraph on *Metallics*).

Drilling by Template.—Owing to the fact, already referred to, that molten metals segregate on cooling, drillings from any one portion of the bar will ordinarily not be representative of that bar. As a consequence, a sample must be taken which will represent all portions of the bar. In copper this is obtained by drilling holes in successive bars by means of a template that covers all, one half, or one-quarter of the bar (the writer's preference being for the last).¹ The template is so arranged that it spaces holes evenly over all, one-half, or one-quarter of the bar and not over 1½ in. apart. While the more durable templates are built of sheet iron, the best material, all things considered, is wood. Flanges, as shown in the illustration of the template, catch both edges of the pig or bar and serve to fix the location of the hole. In the template here illustrated, chalk is rubbed through the hole into the bar. This spot is then marked with a punch, the dust brushed off the bar, and drilling done by the punch mark. With a sheet-iron template, the punch is occasionally driven through the template hole and no chalk used. The drill holes must go completely through the bar, and half the pigs should be drilled from the top and half from the bottom. With lead, as a rule, the punchings are taken on successive bars so that each hole is one-fifth the width of the bar further over and one-fifth further along the length of the bar than was the preceding one. If a single pig of iron is being sampled, the sample should be obtained either over the whole surface of the pig or by drillings taken from several points scattered over the entire surface. In samples of gold and silver bars if they are not taken from a poured sample, the cuttings from the bar should be taken from both top and bottom.

Figures 8 and 9 show the variation in copper bars, Fig. 10 shows what may happen in doré bullion, Fig. 11 gives the segregation in furnace-refined anode cakes, and Fig. 12 that in furnace-refined anodes. The last is interesting as showing that correct results may be obtained by drilling the lugs only.

¹ Theoretically, the quarters are identical, and it takes fewer holes to cover a quarter template than a whole template, spacing the holes on 1¼-in. squares. Consequently, in lots of bullion where the number of the bars are not integral multiples of the number of template holes, a better distribution of holes can usually be attained with quarter templates than with whole, as there will, on the doctrine of chances, ordinarily be fewer holes to scatter, after the even rounds have been drilled.

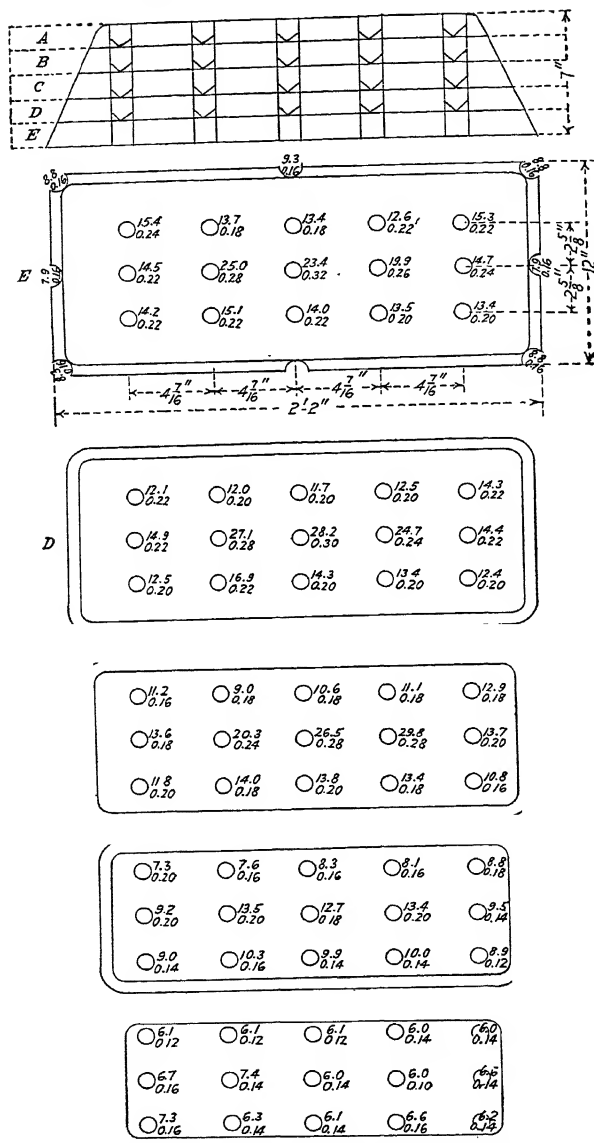


FIG. 8.—Bar of gas-finished converter copper, showing distribution of silver and gold. In each circle the upper figure denotes silver and the lower figure gold, in ounces per ton.

A template should be laid out so that there is not a row of holes on a median line, and, as a rule, less than thirty-five holes to a quarter template will give incorrect results (unless the pigs are small). In making comparisons of different templates and of different methods of sampling copper, it is safe to assume that if the method takes

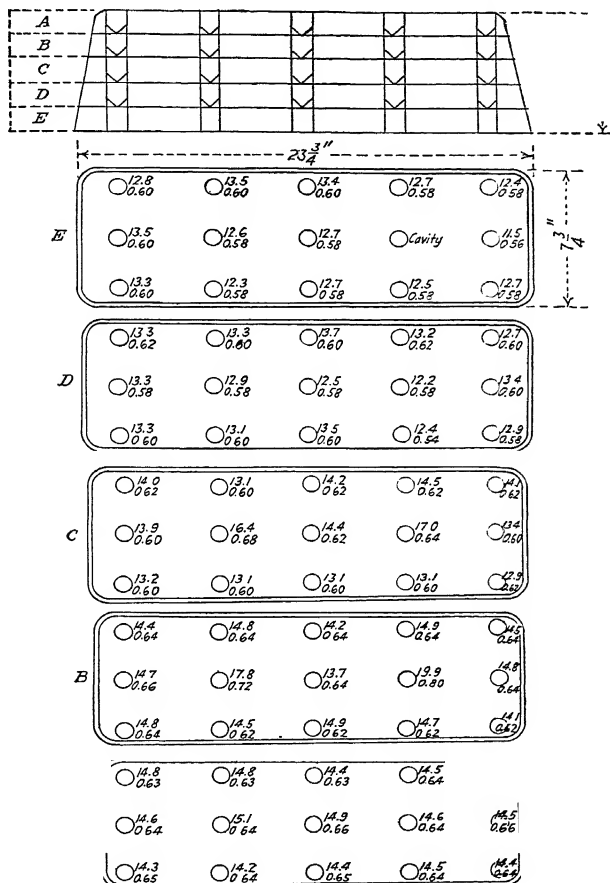


FIG. 9.—Bar of black copper, showing distribution of silver and gold. In each circle the upper figure denotes silver and the lower figure gold, in ounces per ton.

proper care of the silver the results on gold will be correct, as silver has a much greater tendency than gold to segregate.

Theoretically, in any pig having a bevel, if the holes are not spaced to divide the bevel evenly, an error is introduced in template drilling,¹ since the weight of bevel

¹, *Eng. Mining Jour.*, Vol. 92, p. 1173, 1911.

drillings is not proportioned to the weight of the beveled portion of the pig. There is always an error on the corner holes in a beveled pig. These errors are minimized by close spacing of the template holes, and still better, by adopting the plate form.

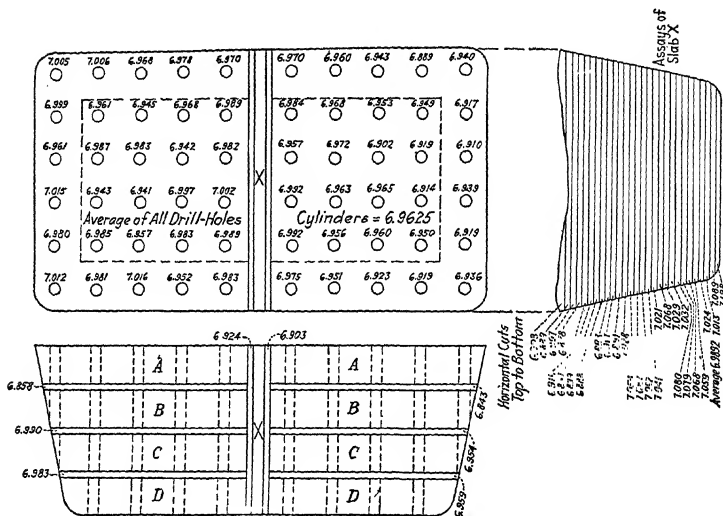


FIG. 10.—Result of assay of doré bullion bar, showing distribution of gold in parts per thousand. All average assays are calculated according to weights of samples.

At any rate, the attempt to have the drill man correct them during drilling usually leads to a worse state of affairs than the one to be corrected.¹

One illustration of incorrect sampling will serve to show how easily an erroneous sample may be obtained.² The material sampled was a low-grade argenteriferous

¹ LIDDELL, *Eng. Mining Jour.*, Vol. 92, p. 1173, 1911.

² KELLER, *loc. cit.*

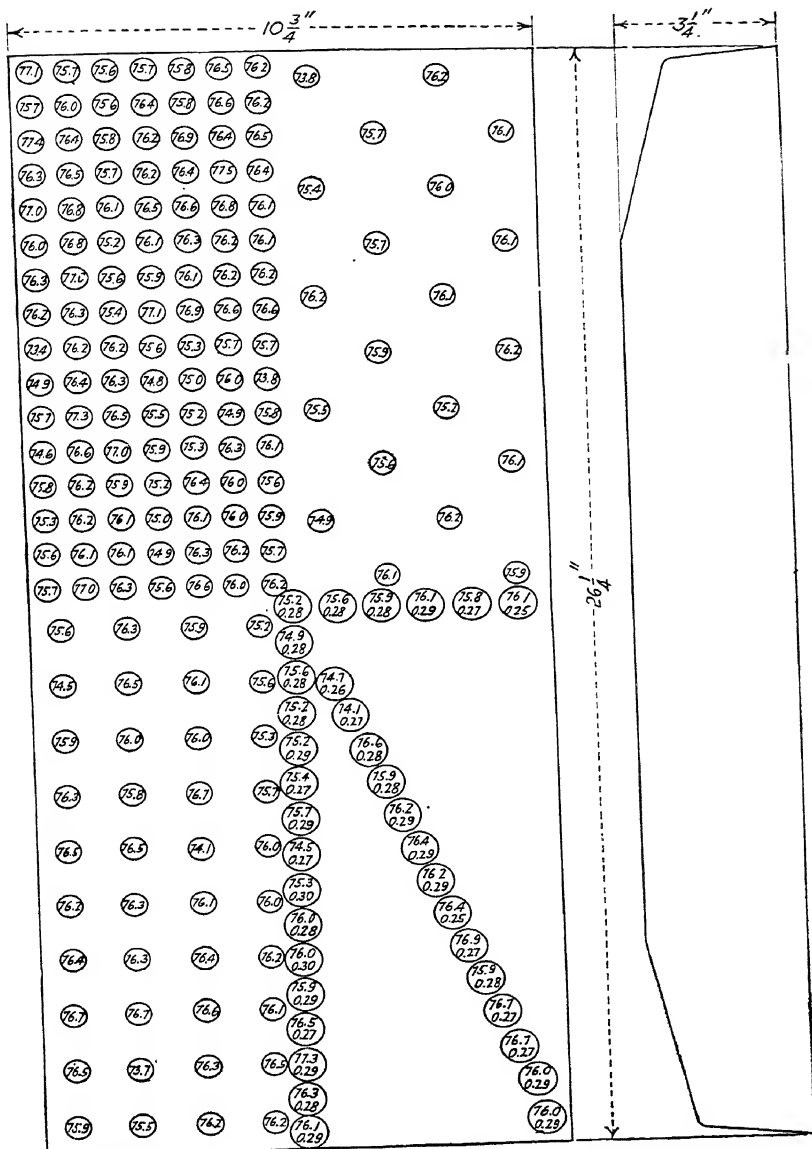
FIG. 11.

Upper left-hand quarter. Average results of assaying quarter, drilled according to 112-hole template, showing silver content in ounces per ton. Average of 112 individual drill-hole samples, 76.04; average of 112 samples combined, 75.50; thin-plate sample, 75.88; shot sample, 75.82.

Lower left-hand quarter. Average results of assaying quarter, drilled according to 40-hole template (the template adopted in practice) showing silver content in ounces per ton. Average of 40 individual drill-hole samples, 75.98; average of 40 samples combined, 75.93; thin-plate sample, 76.26; shot sample, 75.96.

Upper right-hand quarter. Average results of assaying quarter, drilled according to 20-hole template, showing silver content in ounces per ton. Average of 20 individual drill-hole samples, 75.75; average of 20 samples combined, 75.69; thin-plate sample, 76.18; shot sample, 75.88.

Lower right-hand quarter. Average results of assaying quarter drilled according to 35-hole template, showing silver in ounces per ton. Average of 35 individual drill-hole samples, 75.83; average of 35 samples combined, 75.93; average of 16 drill-hole samples taken along longitudinal center line, 75.73; average of 16 drill-hole samples taken along diagonal line, 75.94; average of 6 drill-hole samples taken along lateral center line, 75.78; thin-plate sample, 75.82; shot sample, 75.84. The gold content, in ounces per ton, is also shown for the drill samples of this quarter.



3. 11.—Furnace-refined anode cake. See descriptive legend on opposite page.

copper in the form of cylinders, 14 in. in diameter by 7 in. thick. Previous drill sampling had been performed according to a template with seven holes equally spaced along the diametral line. A new template, which was rational according to the circular area of the cylinders, was laid out in seven zones containing, respectively, from the center outward, 1, 3, 5, 7, 9, 11, and 13 holes, or a total of 49 holes. Forty-nine of the copper cylinders were drilled, making seven rounds for the old template and one for the new. The following results were obtained from 10 assays of each kind of sample:

AVERAGE RESULTS OF TEN ASSAYS OF TWO KINDS OF SAMPLES OF COPPER BULLION CYLINDERS

Templet used in taking sample	Copper, per cent	Silver, per ton, ounces	Gold, per ton, ounces
Diametral.....	97.010	3.1745	0.0215
Rational.....	97.106	2.8585	0.0195
Difference.....	+0.096	-0.3160	-0.0020
Difference, per cent.....	11.05	10.26

General Observations on Copper Drilling.—Drill presses should be braced at the top; taper drills should be used; for a $\frac{1}{2}$ -in. drill, 475 r.p.m. seems to be the right speed for maximum efficiency; a hand feed is superior to an automatic feed; drills should be ground on a drill grinder. On an average, 100 pigs can be drilled after each grinding.

A good method of handling the pigs over the drill tables is shown in Fig. 13. If all the pigs in a lot are being drilled, and the number of pigs is not divisible evenly by the number of holes in the template, the last fractional round must be spaced over the entire template. All edge or all center holes will not do.

It is obvious that no lubricant should be used on the sample drills. It is, however, the custom at some works to use water, or slightly soapy water, to cool the drills as they run. In sampling 10 lots of copper bullion containing over 50 oz. of Ag per ton, lower results were found on both gold and silver when the drills were run wet than when run dry (the drillings were dried before assaying). On low-grade 13-oz. bullion, no differences could be found, outside of small normal swings.

With swiftly driven drills, the drill often drives out a large chunk as it breaks through. These chunks are difficult to grind, and in one works they were being screened out and discarded. This condition is to the buyer's advantage, for on over 100 tests the chunks were of greater value than the fine drillings (by nearly 10 per cent)—independent of whether the drilling was done from the top or the bottom.

Ladle samples taken by abstracting a ladleful of molten metal from the charge and granulating it in the water are often recommended, but the matter of segregation applies to them likewise. No sooner does metal begin to chill in a ladle than segregation begins to take place, and if any skull forms inside a ladle it will ordinarily differ markedly from the material poured out. For this reason the ladle used in sampling any metal should be hot as compared with the metal to be poured from it. The sample should be poured quickly, and no more should be poured after a skull has begun to form in the ladle.

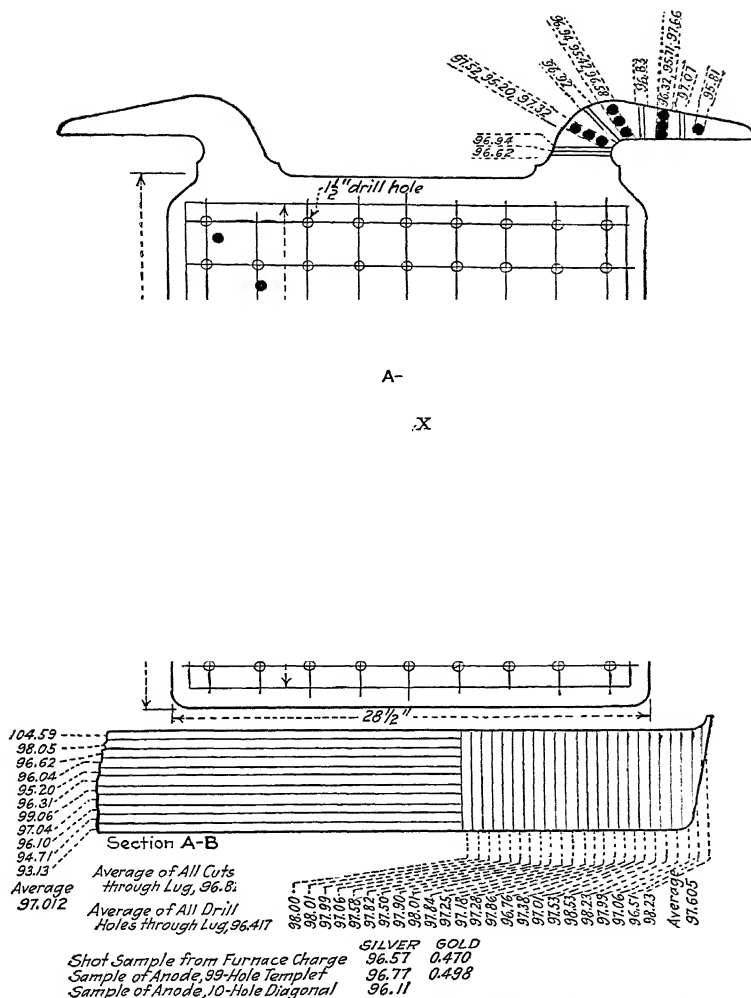


FIG. 12.—Anode of furnace-refined copper bullion, showing distribution of silver and results of sampling.

In experiments made by William Wraith,¹ samples poured from a ladle varied from 75.85 to 81.44 oz. Ag per ton and the ladle skull carried 73.35 oz. The true value was 74.99 oz. Dr. E. Keller gives results obtained on doré, showing again the danger of ladle samples.

¹ *Trans. A.I.M.E.*, Vol. 41, p. 318, 1910.

ANALYSES OF HOT-LADLE AND OF SKULLED-LADLE SAMPLES OF DORÉ BULLION

Lot No.	Hot-ladle samples		Skulled-ladle samples		Difference			
	Assay parts per 1,000		Assay parts per 1,000		Silver		Gold	
	Silver	Gold	Silver	Gold	Over	Under	Over	Under
1	973.79	11.21	977.62	10.78	3.83	J.43
2	981.56	10.84	977.32	10.78	4.24	0.06
3	977.19	9.71	976.32	8.98	0.87	0.73
4	981.12	8.18	973.46	7.84	7.66	0.34
5	976.89	8.21	980.13	8.17	3.24	0.04
6	966.60	10.40	976.88	10.42	10.28	0.02
7	977.72	9.58	976.93	9.27	0.79	0.31
8	976.92	11.18	976.80	10.60	0.12	0.58
9	979.13	10.37	979.30	10.30	0.17	0.07
10	976.00	10.40	976.71	9.89	0.91	0.51
Total	18.42	13.68	0.02	3.07
Mean difference per lot	0.47	0.31

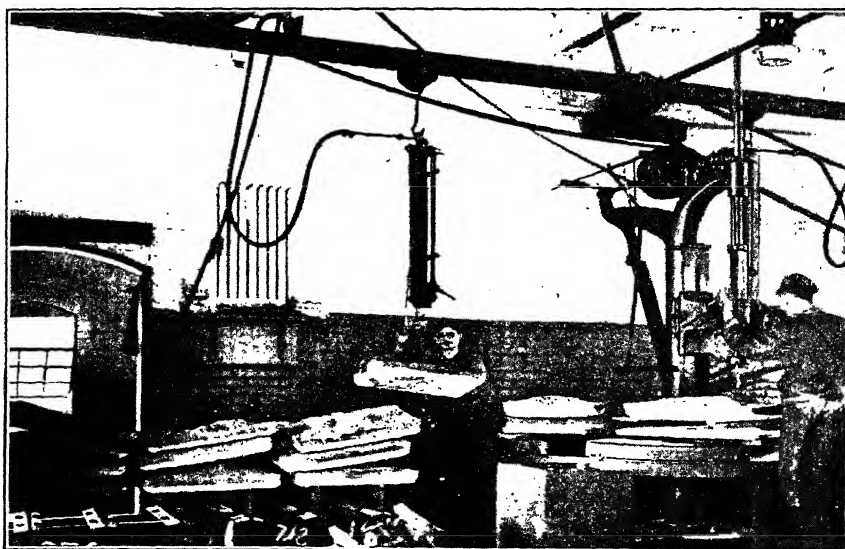


FIG. 13.—Method of handling pigs over drill tables.

In some of the writer's experiments cakes were poured from the ladles and these cakes sampled by template in the usual manner, and the ladle skull also sampled.

	Ag ounces per ton		Au ounces per ton	
	Skull	Cake	Skull	Cake
Experiment I.....	150.68	144.08	3.67	3.62
Experiment II.....	146.48	144.60	3.67	3.55
Experiment III.....	149.52	137.86	3.63	3.54
Experiment IV.....	145.69	138.96	3.61	3.44
Experiment V.....	430.70	388.09	7.06(!)	7.50(!)

Lead Sampling.—As can be seen from the character of the metal, lead can be sampled with less likelihood of error than can any other metal. The process is thus described by Raht:¹

"After melting the bullion bars down in a kettle holding about 20 tons, the coppery dross is skimmed off and sent back to the blast furnace to be smelted over; with this one of the worst sources of error in sampling is removed. The bullion is then thoroughly stirred, say, for 5 min., and while this is still going on a number of samples are taken by means of a small ladle holding approximately one-half an assay ton of metal. It is important that this ladle should be at least as hot as the bullion, so that none of the metal would stick to its surface, otherwise the samples would be too low in silver. The samples thus obtained are taken direct for assaying without any trimming or clipping. Instead of trimming the sample down to the exact weight of half an assay ton, the exact weight of each sample is ascertained, whatever it may be, and from the resulting silver button the contents are figured. Care is taken, however, in the selection of the samples for assaying out of the great number originally taken that these should be somewhere near the same weight, so that when they are cupeled side by side they would finish about the same time.

"It will be seen that all the objections to punch sampling are met in this method. Still, another one might be raised against it, which is the uncertainty whether the mixture was thorough. I must say that this does not appear to carry much weight, but it can be overcome by taking samples at regular intervals out of the stream of metal running from the kettle into the molds. These samples would have to be remelted in a graphite crucible and out of this again one-half assay ton samples taken, which are then treated as above. This method, of course, brings in again the objection to the remelting. There cannot be the least doubt that either of these two dip samples would be nearly absolutely correct, but they can be carried on only at the works of the seller of the bullion, as it would be next to impossible to obtain a correct sample from the dross skimmings. It is to be hoped that the dip sample will be generally introduced wherever practicable, and so do away with the annoying sample and assay differences between the bullion producer and refiner."

In discussing the lead-smelting practice at the El Paso Smelting Works, El Paso, Tex., H. F. Easter describes² the sampling of the base bullion as follows:

"Whether drossed at the blast furnace or at the remelting furnace, the bullion is tapped into a cast-iron cooling pot. The dross is skimmed off carefully and thor-

¹ RAHT, The Distribution of Silver in Lead Bullion. and the Different Methods of Sampling, *Mineral Ind.*, Vol. 3, p. 414, 1894.

² Excerpt from a paper entitled "Lead Smelting at El Paso," presented at the San Francisco meeting of the A.I.M.E., September, 1915.

	Coarse, 173 g., per cent	Fine, 16 g., per cent	Calculated composition, per cent
Total carbon.....	2.62	11.66	3.39
Graphite.....	2.38	10.91	3.10
Combined carbon.....	0.24	0.75	0.29
Silicon.....	2.44	2.27	2.42
Phosphorus.....	0.51	0.56	0.515
Sulphur.....	0.03	0.073	0.079

Another case (taken from the same source) was a Swedish gray iron which was planed over the entire cross section, giving 693 g. of shavings. These were screened so as to obtain three portions: on 80 mesh; through 80 mesh and on 120 mesh; and through 120 mesh. Analysis on the screened portions gave results as follows:

	Coarse, 469 g., per cent	Medium, 193 g., per cent	Fine, 31 g., per cent	Calculated analysis, per cent
Total carbon.....	3.53	4.04	22.00	4.50
Graphite.....	2.12	2.53	20.94	3.07
Combined carbon.....	1.41	1.51	1.06	1.43
Silicon.....	0.81	0.83	0.76	0.80
Phosphorus.....	0.051	0.044	0.03	0.048
Sulphur.....	0.017	0.014	0.013	0.016

On a sample of comparatively clean converter copper, screened over a 12-mesh screen, the portion on the screen assayed 98.94 per cent Cu, that which passed the screen, 97.21 per cent. The proportions were as 24:1.

There is no need of multiplying instances with analyses of other materials. The sampler as well as the analyst must realize the variations that may exist between the coarse and fines of a sample, and the sampler must so arrange his methods that the correct proportions of each may be turned into the laboratory. The methods by which the analyst will treat the sample after it is delivered to him do not fall within the scope of this book.

Hardened steels may be annealed before machining. Annealing for 15 min. at 750 to 800°C., followed by slow cooling until the pearlite point is reached (700°C.), is sufficient to anneal carbon steels. Below 600°C. the steel may be cooled as quickly as desired. In certain alloys and self-hardening steels, it will be necessary to use chrome-tungsten and other alloy steels for the planing tool. With material that cannot be worked even with special steels, it is necessary to knock off small pieces from different parts of the sample and to pulverize these pieces. It is recommended by Deiss and Bauer that the sample be wrapped in heavy linen cloth before pulverizing and the broken pieces be carefully cleaned to free them from linen fibers. It should be noted that in cases of this class carbon analyses are always subject to some doubt, as it is next to impossible to remove all the cloth.

As a very elementary precaution, in taking samples of iron or steel it is well to look at the drills or other cutting tools, to see that no large pieces have been broken out of

them. If there have been, these pieces can usually be found and taken out of the samples.

METALLICS

The limiting case of variation between coarse and fine material in the same sample is that of metallic particles or masses occurring in material which otherwise is capable of being crushed. In this case the sampler should weigh the metallic and nonmetallic portions of the sample as soon as they can be properly separated, sample each separately, and report the original weights to the laboratory.

Metallic substances often occur that can be ground by suitable means and the powder added to the pulp sample. This, however, is a questionable procedure, for this powder must be of different composition from the easily pulverized material and its introduction into the sample makes a "spotty" sample. Much of the trouble, for instance, with samples of gold ore that give discordant results is due to forcing through the sampling screen small metallic particles that should have been removed and weighed and reported as "metallics."

It may be noted at this point that the formula for computing the average value of a sample containing metallics is as follows:

If W = weight of pulp.
 w = weight of metallics.
 A = assay of pulp.
 a = assay of metallics.

$$\text{Then average value } V = \frac{W \cdot A}{W + w}$$

Occasionally, as in sampling reverberatory slags, metallics will be taken out at each stage of the crushing. In this case a note should be made of the relative weights of metallic and nonmetallic portions at each stage, but apparently no great error results if but one sample of metallics is turned in along with the final sample of pulp.

The assay of the "fine" of the next to the last stage must be calculated from the assays and weights of the coarse and fine of the last stage. The assay of the coarse of the next to last stage is assumed to be the same as the assay of the coarse of the last stage, and so on.

In the general sampling of slags and other fused products, it should be noted that these materials will decompose most easily when they have been suddenly chilled, and it is, therefore, recommended, for the convenience of the analyst, that this be done when possible, in sampling fused masses.

SAMPLING OF LIQUIDS

Concerning the sampling of liquids or semiliquid material, there is not much to say except to point out the necessity of rapid agitation to ensure uniformity of the bulk to be sampled and drawing off the sample during the agitation or so quickly thereafter that segregation has not had time to take place.

The same precautions are necessary in further laboratory tests of these samples. Even though the liquids contain no sediment, it must still be remembered that temperature changes in the room in which a liquid is standing are sufficient to cause differences in concentration of dissolved solids between the top and the bottom layers.

Errors are sometimes introduced when samples are taken at high temperatures and crystallization occurs, because the samples are cooled before they are used.

No general directions can be given upon the proper treatment of samples where there is a sediment, *i.e.*, how to estimate correctly the influence of the sediment on

the original sample and to what extent the sediment should be washed and what is the influence of the washing water upon the sample.¹ This matter must be determined by the chemist for each case independently. The general instruction can, however, be taken to heart that, where these samples are not routine matters and the procedure absolutely worked out, it will invariably pay to recheck these calculations after making them, or better still, to attempt to deduce the final results by an entirely new chain of reasoning.

Most of the automatic pulp samplers depend upon a water- or pulp-driven tipping device which drags a cutter through the stream of outgoing pulp. It seems strange that a simple tipping box one side of which discharged to the sample and one side to waste, with a much smaller mass required to discharge to the sample vat, is not sufficient, but all the pulp samplers the writer has seen have been complicated affairs.

SAMPLING GASES

In **sampling gases** it is again difficult to give general directions. The use of two bottles connected by a rubber tube attached to glass or metal tubes reaching to the bottoms of the bottles and passing through two-hole corks, the other aperture being closed by another glass tube running to a rubber tube and stopcock, as an aspirating apparatus is well known. The apparatus is simple, but has the grave defect that the water over which the gas is usually collected absorbs the easily soluble gases, such as CO_2 , NH_3 , and SO_2 . This can be obviated to some extent by trying to saturate the water in the bottles with the gas to be collected, but there still is danger either that the water will not be nearly enough saturated to stop absorption or that it will contain so much gas that it will be in unstable equilibrium and will be given off into the sample.

To a certain extent these errors can be avoided by the use of a 10 per cent sulphuric acid solution, as the solubility of most gases (except bases) is less in this than it is in water. The errors can only be entirely done away with, however, by collection over mercury, or by the use of previously exhausted vacuum tubes, the value of which can be opened in the gas to be sampled.

In sampling furnace gases, the results are almost invariably vitiated by a rush of air along the sampling tube down into the zone to be sampled. Leaky points in the apparatus should always be carefully watched for also.

INTENTIONAL VITIATION OF SAMPLES

Salting.—Of almost equal importance with knowing what to do to obtain correct samples is to know what may be intentionally done to render samples incorrect. This is, of course, of most importance to the mining engineer who is sampling mines, or to those engaged generally in sampling for the precious metals, but the sampler of any sort of material should know what may be done to render his work misleading.

Many of the following methods are taken from a paper by George A. James on the subject.²

Salting may, of course, be innocently or criminally done. Among the ways in which the latter may occur is the use of old coin bags for carrying samples in. The abraded particles left clinging to the bags may cause serious errors. Another way of salting one's own samples is to use one sack for two samples, separating them by a string around the center of the bag. In such a case the samples must be removed by

¹ The classic tables bearing on samples and problems involving a pulp suspended in a liquid are those of Clevenger, Young, and Turner, *Eng. Mining Jour.*, Dec. 19, 1924. This article should be referred to if many such samples are to be taken and assayed.

² *Mining Eng. World*, May 20, 1912, p. 1099.

cutting both ends of the sack, leaving the string tied. Inferior or porous sacks must be avoided, or else the fines may be lost through the meshes of the bag. Canvas used for catching the sample as broken in the mine may be contaminated, as may also the tools. This last remark applies with equal force to everything used throughout the sampling process. Proper allowance is not usually made for the pulp that may be retained by the grinding machinery, nor for metallics that may stick to the muller, bucking board, etc.

As a rule, assayers do not give sufficient attention to drying the samples properly. While the failure to consider 2 to 3 per cent of water may at times be a wise factor of safety, omission of proper drying may sometimes lead to disastrous errors.

The use of the precaution commonly taken to overcome possible salting, washing the ore with water, is not so efficient as is usually supposed. While gold chloride and silver nitrate are both soluble, they are, after drying on the ore, usually reduced and rendered insoluble. Seals are of little value, as a whole wax seal can be raised with a knife, and later replaced intact. Cloth or paper bags yield to the hypodermic syringe, while even glass bottles will succumb to a hypodermic needle thrust through the cork.

Salting in precious metal samples may be done by use of gold under the fingernails, or in the hair or beard, or by cigarette or pipe ashes carelessly (?) dropped into the sample. A case is on record where salting was done by gold dust in clay pellets "carelessly" shot at the sample pile.

A method of salting practically impossible to detect, or rather a method of influencing the sampling, is known as "drawing the center." In this the men who are piling up the cone purposely place the successive shovelfuls a little to one side of the apex (always moving the new apex in the same direction). Eventually, this gives a pile with the original apex of the cone well to one side of the new and buried in the cone. The quarter containing the original apex will then contain an undue proportion of fine, and if this quarter and its opposite are taken the sample contains too much fine; if the adjacent quarter and its opposite are taken, the sample contains too much coarse. These quarters will be either taken or rejected according to which way the sample is to be thrown. A way of avoiding this is to place a small stick upright in the center of the cone as soon as it is well started and see that the stick is always kept in the center of the heap.

In general, in watching any sampling by the coning and quartering methods it is advisable continually to compare the looks of the rejected heap with the accepted, and if it does not look right, repile the entire sample. In general, it should be remembered that in coning nonhomogeneous material no mixing takes place, but rather a classification, due to the large pieces rolling to the outside of the pile. Coning and quartering are not means of mixing, but only of dividing a sample.

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CHAPTER V

SCREENING AND SIZING

BY ALBERT E. REED¹

Screening is required in nonferrous metallurgy in connection with crushing and grinding operations and also in the preparation of products of specified size for further treatment or for a particular industrial application. The last two decades have seen a revolutionary advance in the screening art and a rapid adoption of the newer forms of screening equipment by industry.

The types of screens in use in the metallurgical field in 1915 were basically the same as those used in 1556 when Agricola was writing his "De re metallica." They were the stationary bar grizzly, the revolving screen, and the slow-speed shaking screen.

The development of the porphyry-copper and other low-grade ores created an insistent demand for more efficient means of handling huge tonnages of materials and reducing the run-of-mine material to size required for concentration, flotation, or leaching at a fraction of previous unit costs. New types of primary and secondary crushers were developed and improved, while ball milling came into its own in fine grinding.

It was soon found that these crushing and grinding processes could not achieve their full capacity and efficiency and consequently the lowest unit cost of reduction without the aid of screening.

The need for a more efficient means of size separation was so urgent that a new conception of screening was soon created and gradually so perfected through the entire range of sizing practice that today the stationary grizzly, the trommel, and the slow-moving shaker screen are practically relics of the past.

This revolution was brought about by what is known as the "high-speed vibrating screen." Its first great impetus, starting in 1917, was given by the Hum-mer electric screen developed for separations finer than $\frac{1}{2}$ in. and was followed in the late 1920's by the high-speed circle-throw screen which set the pace for other forms of mechanical screens that have since been developed and placed on the market.

Theory and Purpose of Screening.—Screening is the art of separating solid materials according to particle size. This is accomplished on a commercial scale by means of a surface containing a large number of openings of the required size. Woven wire screen is the screening medium most commonly used as it combines strength with the greatest number of openings per unit of area, but parallel bars and perforated plate are also used for certain specific coarse-screening applications. A screening medium is a multiple go or no-go gauge, the object of which is to measure an infinitely large number of particles in the briefest possible interval of time. In all screening processes in general use, gravitation is an important factor in accomplishing the desired separation.

In the simplest process of screening, the material to be graded is placed on a flat screening surface. Those particles smaller than the openings in the screen will fall through and be accepted as "undersize," while those particles which are larger than the openings will be rejected as "oversize."

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A complete separation will occur, considering the screening medium as without movement, only when the particles on the surface are distributed "one particle deep"; otherwise the larger particles might prevent many small particles from reaching a position on the screening medium where they can be measured and accepted.

Such a slow process of particle measurement would be unthinkable in modern industry where high tonnages are often screened again and again with each size reduction.

To speed up the process of size measurement, it is evident that a continuous agitation of the material to be screened is required so that all particles which might pass may be brought into contact with the screening surface and given an opportunity to pass its openings. To provide this agitation or sorting action and to keep the openings free from blinding is the purpose of screening machinery. It is evident that the machine that will, in a unit of time, complete the size measurement of the largest number of individual particles per unit of area is the most efficient.

Screening Media.—Screening media may be grouped into three classifications—bar or rod screening (including wedge wire), perforated plate, and woven wire cloth.

Bar or rod screens made from steel, cast iron, or alloy bars have long been used for grizzlies or stationary scalping screens, and modifications of these types are in use for coarse sizing on vibrating screens. Many specially shaped bars are used, the most popular being wedge shaped to reduce clogging of the screen.

Wedge wire is a fine-opening edition of the bar screen, of limited application, because of its small percentage of open area, excessive weight, and high cost per unit of area. Its best present application is as a stationary dewatering screen.

Perforated metal screens are made in a variety of metals, including heat-treated alloy steel. Holes, while generally round, can also be made in other shapes such as oval, square, hexagonal, and straight or diagonal slots.

Perforated plate is used on many types of trommels, especially where the structural stiffness of the plate is a required part of the trommel design.

TABLE 1.—APPROXIMATE ROUND AND SQUARE OPENING EQUIVALENTS FOR TESTING SIEVES¹

Square openings, in. □	Round openings, in. ○	Square openings, in. □	Round openings, in. ○
4	4 $\frac{3}{4}$	1 $\frac{1}{4}$	1 $\frac{1}{2}$
3 $\frac{3}{4}$	4 $\frac{1}{2}$	1 $\frac{1}{8}$	1 $\frac{3}{8}$
3 $\frac{1}{2}$	4 $\frac{1}{4}$	1	1 $\frac{1}{4}$
3 $\frac{5}{16}$	4	$\frac{7}{8}$	1
3 $\frac{1}{8}$	3 $\frac{3}{4}$	$\frac{3}{4}$	$\frac{7}{8}$
3	3 $\frac{1}{2}$	$\frac{5}{8}$	$\frac{3}{4}$
2 $\frac{3}{4}$	3 $\frac{1}{4}$	$\frac{1}{2}$	$\frac{5}{8}$
2 $\frac{1}{2}$	3	$\frac{3}{8}$	$\frac{1}{2}$
2 $\frac{1}{4}$	2 $\frac{3}{4}$	$\frac{5}{16}$	$\frac{3}{8}$
2 $\frac{1}{8}$	2 $\frac{1}{2}$	$\frac{1}{4}$	$\frac{5}{16}$
2	2 $\frac{3}{8}$	$\frac{3}{16}$	$\frac{1}{4}$
1 $\frac{7}{8}$	2 $\frac{1}{4}$	$\frac{5}{32}$	$\frac{3}{16}$
1 $\frac{3}{4}$	2	$\frac{3}{32}$	$\frac{1}{8}$
1 $\frac{1}{2}$	1 $\frac{3}{4}$		

¹ Adapted from Table 5, "Coarse Aggregates," *Simplified Practice Bulletin* R163-36, United States Department of Commerce.

The principal drawback to perforated plate is its generally low percentage of open area as compared with that of wire cloth. Increased open area can be obtained with square punched holes, but at a sacrifice in strength. When screening abrasive materials, the wear tends to change the size of the openings, particularly in the case of the finer perforations.

Various investigators have demonstrated that for the average crushed material the maximum size of particles passing a round hole is 81 per cent of those passing a square hole of equivalent side dimensions as the diameter of the round hole. (For comparative round and square openings see Table 1.)

Woven wire cloth is the most versatile and widely used of all screening media. It can be woven in practically any metal, and the available specifications are infinite. One manufacturer lists over 7000 combinations of opening, wire diameter, and metal. Woven wire cloth for screening media is obtainable in three general types of openings: (1) square mesh; (2) oblong opening (Ton-Cap, Rek-Tang, etc.); (3) long-slot opening (Non-Blind, Ty-Rod, etc.).

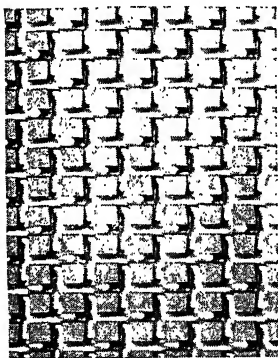


FIG. 1.—Screen, $\frac{1}{8}$ -in. opening, 0.072-in. wire.



FIG. 2.—No. 770 Ton-Cap screen.

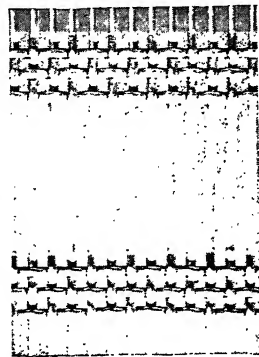


FIG. 3.—No. 936 Ty-Rod.

Square-opening wire cloth is generally woven by the principal manufacturers from 4-in. opening, woven with 1-in. rods, to 400 mesh (160,000 holes per square inch). Each mesh or opening down to 100 mesh is woven in 5 to 20 different wire diameters, so that a user has wide choice between the extremes of durability and capacity.

Space Cloth.—Coarse screens are specified by the width of opening between the wires and are known as space cloth—the standard range being between 4-in. opening and $\frac{1}{8}$ in. The standard list of square-mesh screens in Table 2 shows the grades recommended for various types of screening machines.

The *oblong-slot screen* (Fig. 2) was invented to give greater discharge or screening area while maintaining, or even increasing, the durability. In other words, the greater open area of the oblong opening is designed to let the undersize through instead of holding it back with blank metal. A screening station can often be the bottleneck of an entire plant's capacity. Many instances have been reported where merely changing a screening surface from a round or square opening to an oblong slot, with no other change, has resulted in a 10 to 20 per cent increase in *total* plant capacity.

Some prejudice has existed in the past against oblong-opening screens on the theory that their use would result in slivery or flaky particles passing the screen. The fallacy of this argument is that when a crushed material contains elongated or flaky particles

TABLE 2.—STANDARD COARSE SCREEN SPECIFICATIONS

Open- ing, in.	Wire diameters							
	Medium light		Medium		Medium heavy		Heavy	
	Wire diam- eter, in.	Per cent open area	Wire diam- eter, in.	Per cent open area	Wire diam- eter, in.	Per cent open area	Wire diam- eter, in.	Per cent open area
4 ^a	0.500	79.0	0.625	74.8	0.750	70.9	1.000	64.0
3½ ^a	0.4375	79.0	0.500	76.6	0.625	72.0	0.750	67.8
3	0.4375	76.2	0.500	73.5	0.625	68.5	0.750	64.0
2¾	0.375	77.4	0.4375	74.4	0.500	71.6	0.625	66.4
2½	0.375	75.6	0.4375	72.4	0.500	69.4	0.625	64.0
2¼	0.375	73.4	0.4375	70.1	0.500	66.9	0.625	61.2
2	0.3125	74.8	0.375	70.9	0.4375	67.3	0.500	64.0
1¾	0.3125	71.9	0.375	67.8	0.4375	64.0	0.500	60.5
1½	0.250	73.4	0.3125	68.5	0.375	64.0	0.4375	59.9
1⅜	0.250	71.5	0.3125	66.5	0.375	61.6	0.4375	57.5
1¼	0.250	69.4	0.3125	64.0	0.375	59.2	0.4375	54.8
1⅓	0.225	69.6	0.250	67.0	0.3125	61.0	0.375	55.7
1	0.225	66.6	0.250	64.0	0.3125	58.0	0.375	52.9
⅞	0.207	65.3	0.225	63.3	0.250	60.5	0.3125	54.3
¾	0.192	63.4	0.207	61.4	0.250	56.3	0.3125	49.8
⅝	0.177	60.7	0.192	58.5	0.225	54.0	0.250	51.0
½	0.162	57.1	0.177	54.5	0.192	52.2	0.207	49.8
⅙	0.148	55.8	0.162	53.2	0.177	50.7	0.192	48.3
⅓	0.135	54.1	0.148	51.4	0.162	48.7	0.177	46.1
⅕	0.120	52.2	0.135	48.8	0.148	46.0	0.162	43.4
¼	0.105	49.6	0.120	45.6	0.135	42.2	0.148	39.4
⅓	0.080	49.1	0.092	45.1	0.120	37.2	0.135	33.8
⅓	0.054	48.7	0.072	40.2	0.092	33.4	0.105	29.5

^a Recommended by the Division of Simplified Practices, U.S. Department of Commerce, for screening of mineral aggregates.

^a These specifications are not shown in simplified practice recommendations, but because of their wide use, are included in table.

Heavy wire recommended for trommels.

Medium heavy for high-speed vibrating and shaking screens.

Medium light and medium for other vibrating screens.

these shapes will in all probability occur in the entire size range. In such a situation a square-mesh screen, of say ½-in. opening, will obviously retain many particles that would pass through a ½- × ⅓-in. oblong opening. In other words, the undersize of the square opening will still contain the "slivers and flakes," but they will be of a smaller average size. The answer is the selection of a sufficiently finer oblong opening to produce the same equivalent size to the square mesh. In spite of having to use the smaller opening, there will still be the advantage of greater discharge area, cleaner oversize, and less tendency to blind.

Long-slot Screens.—Woven screens with extra-long slots (Fig. 3) were designed to handle damp and sticky materials that would tend to blind square or even oblong (short-slot) screens.

Screening Machines.—As indicated at the beginning of this chapter, the past fifteen years have witnessed a revolution in the screening art. While many obsolete

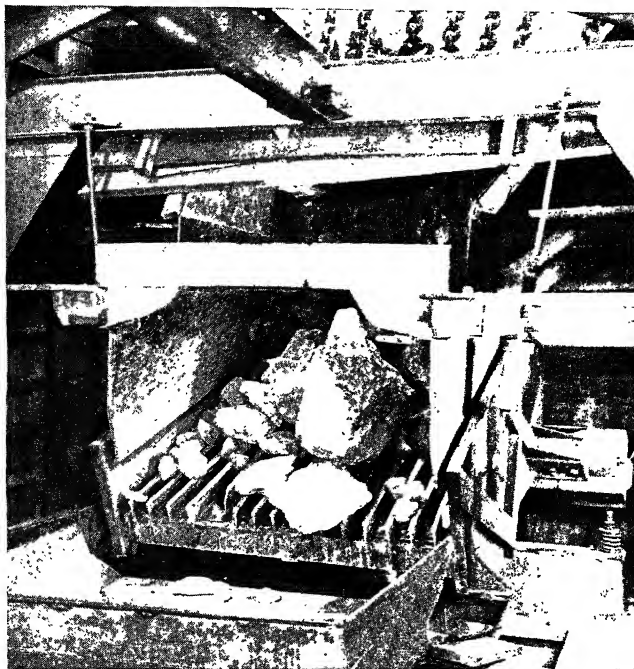


FIG. 4.—Electrically vibrated feeder and grizzly.

types of screening devices will still be found in use, new installations will generally contain one or more of the following types:

1. Grizzlies, stationary or vibrated.
2. Mechanically vibrated screens.
 - a. Circle throw, positive drive.
 - b. Circle throw, unbalanced-weight drive.
 - c. Reciprocating motion, eccentric drive.
 - d. Reciprocating motion, unbalanced-weight drive.
 - e. Reciprocating motion, cam operated.
3. Electrically vibrated screens.

Stationary Grizzly.—The ordinary grizzly is the simplest of all separating devices, but can be used only for very rough primary scalping. It consists of spaced bars or rails and is generally used for removing "fines" from primary crusher feed.

Cantilever Grizzly.—This type of grizzly is fixed at one end and so designed that the impact of the feed will give some movement or vibration to the bars. When made of wedge-shaped bar, this type has an obvious though limited advantage over a purely stationary type.

Vibrating Grizzlies.—Several manufacturers of circle-throw positive-drive screens have designed super-heavy-duty models for heavy-tonnage primary screening. The deck is usually made from spaced rods, although wedge bars and perforated plate are also used. These units operate with $\frac{1}{4}$ - to $\frac{1}{2}$ -in. circle throw, at speeds of 850 to 1200 r.p.m. Several makes of electrically vibrated grizzlies are also available, some (as in Fig. 4) combining both vibrated feeder and grizzly in one machine.

Vibrating Screens.—The principal types of vibrating screens being used in the field of nonferrous metallurgy at the present time for coarse, medium, and fine separations can be described by groups as follows:

High-speed Fixed Eccentric-shaft Circle-throw Screen.—This screen is characterized by an eccentric shaft rigidly mounted on the base frame of the machine and utilizes four bearings. It has been a very popular type for coarse and medium screening and

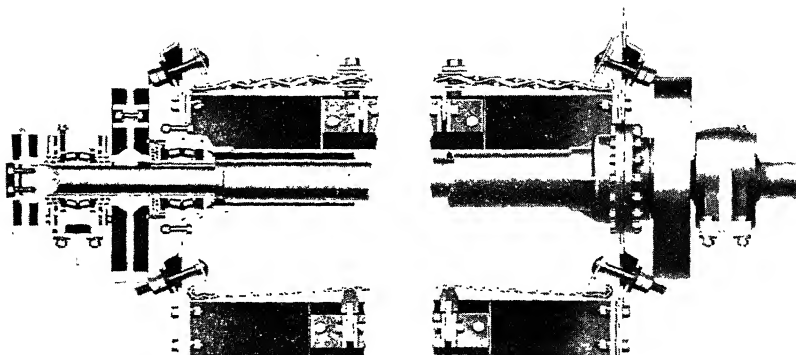


Fig. 5.—High-speed fixed-throw eccentric shaft. (Courtesy of Tyler-Niagara.)

is available in a number of makes such as Allis-Chalmers, Gyrex, Link-Belt, Tyler-Niagara. A typical cross section through the shaft assembly is shown in Fig. 5.

High-speed Full-floating Balanced Eccentric-shaft Circle-throw Screen.—This screen is characterized by a full floating eccentric shaft supported on the base frame of the machine by live-rubber shear-type mountings, and also utilizes four bearings. It operates on a unique principle whereby the shaft is so balanced that it rotates around its own natural center of rotation lying between the two sets of bearings, and all the forces acting on the base frame of the machine are balanced by equal and opposite forces created by the same rotation of the shaft. The live-rubber mountings also absorb the impact shock from the material passing over the screening surfaces.

This screen, known as the Ty-Rock (Fig. 6), is suitable for both horizontal and low-angle operation, the positive action assuring a forward movement of the material whether operated flat or at an angle.

Rotating Unbalanced-weight Screens.—In screens of this type, the screen body is spring supported or suspended and the motion imparted by means of revolving weights on a two-bearing shaft mounted on the screen body. This general type is made by a number of manufacturers, some of the trade names being Aero-Vibe, Overstrom,

Plato, Ripl-Flo, Universal, Vibrex. A typical model of this last-named type is shown in Fig. 7. This type is generally confined to light-tonnage work. Link-Belt puts out a model especially designed for wet screening.

High-speed Eccentrically Driven Reciprocating Screen.—This type of screen is characterized by an eccentric shaft connected to the vibrating body by means of a connecting-rod arrangement. A well-known design of this construction incorporates an eccentric shaft mounted on a balancing frame and connected to the vibrating body by means of a connecting rod in the form of leaf springs. Both the screen body and the balancing frame are mounted at the end of upwardly extending leaf springs which

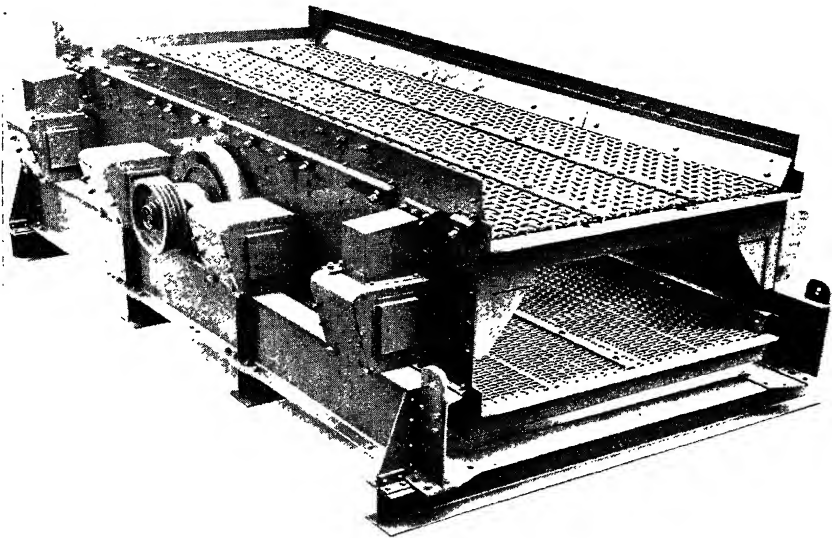


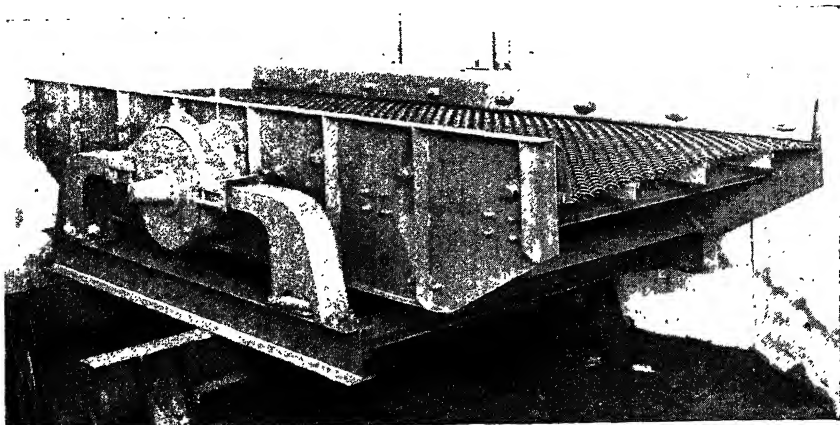
FIG. 6.—Ty-Rock eccentric-shaft circle-throw screen.

permit the oscillatory reciprocating motion. This type of screen is particularly adapted for horizontal operation and is illustrated by the Symons screen (Fig. 8).

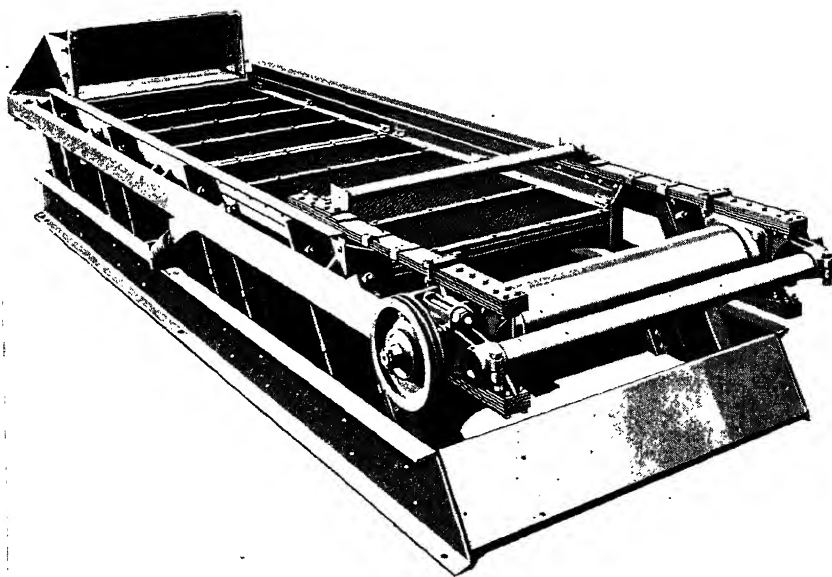
Reciprocating-motion Unbalanced-weight Drive.—Variations of the usual unbalanced-weight type are the Allis-Chalmers Low-Head and the Robins Eliptex. The Low-Head (Fig. 9) operates horizontally. The body is spring suspended, and the action is obtained by a system of oppositely revolving counterweights that are so arranged to produce a reciprocating vibratory motion that gives the forward motion to the material being screened.

Electrically Vibrated Screens.—These screens are generally more economical to operate per unit of area and, within their range, will be less subject to blinding than mechanical screens.

In this type of screen the vibration is produced by means of electromagnets operated by pulsating current, the source of which may be a low-frequency generator set or an electronic rectifier unit. In the Hum-mer electric screen (Fig. 10), the upward stroke of the vibrator is brought to a sudden stop by means of a striking block which



7.—Vibrex screen. (*Courtesy of Robins Conveyors, Passaic, N.J.*)



g. 8.—Symons eccentrically driven reciprocating screen. (*Courtesy of Nordberg Mfg. Co., Milwaukee, Wis.*)

imparts a sharp vibration to the screening surface. These screens can be operated either wet or dry. Electric screens are usually supplied in 3×5 ft. or 4×5 ft. units. Tandem arrangements are often employed to give double the length of travel to the material for close sizings. Other types of electrically vibrated screens on the market are the Jeffrey-Traylor, Tyler Type 400, and the Utah.

Screens for Closed-circuit Grinding.—Electrically vibrated screens are an ideal means of closing the circuit in wet-ore grinding operations. They have proved their ability to handle large tonnages as fine as 65 mesh, using stainless-steel Ton-Cap cloth as a screening medium.

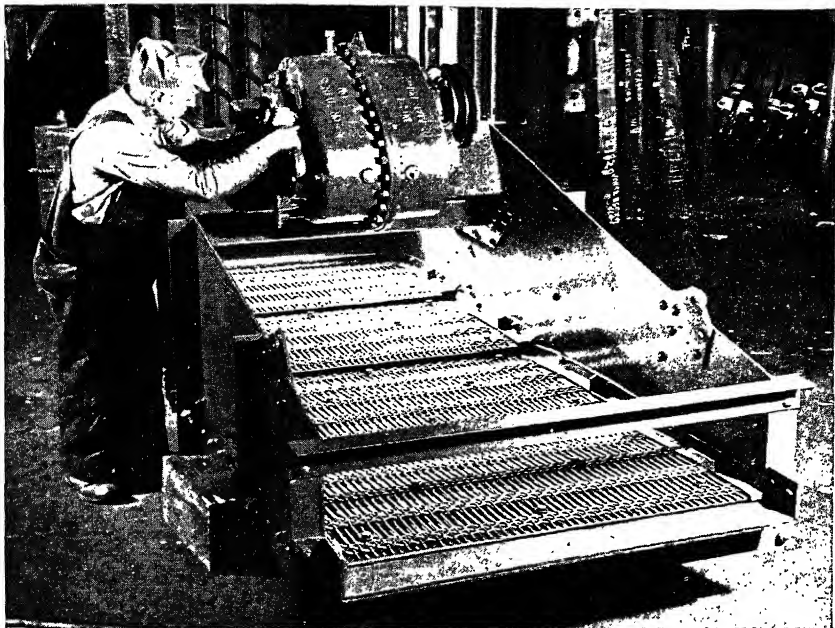


FIG. 9.—Double-deck sink-float Low-Head vibrating screen. (Courtesy of Allis-Chalmers Mfg. Co., Milwaukee, Wis.)

Since screens will make the separation according to particle size regardless of specific gravity, they are more efficient for closed-circuit grinding work than gravity classifiers. It is also characteristic of screen circuits that there is more finished product removed from the return to the grinder than when classifiers are used, thus reducing the circulating load and making possible increased ball-mill capacity.

The finished product of a screen circuit will contain less "slimes" than classifier circuits and is thus often highly advantageous in subsequent concentration processes, particularly when treating tin and tungsten ores. A typical closed-circuit grinding arrangement is shown in Fig. 11.

Screens in Heavy-medium Separation Plants.—Concentration by the heavy-medium, or sink-and-float, separation process is finding a rapid acceptance in the field of nonferrous metallurgy. In this process the raw ore is first screened to a definite

size range (*i.e.*, minus $1\frac{1}{2}$ plus $\frac{1}{4}$ in.). After the separation has been completed in the heavy-medium cone, both the "float" and the "sink" products must be drained and

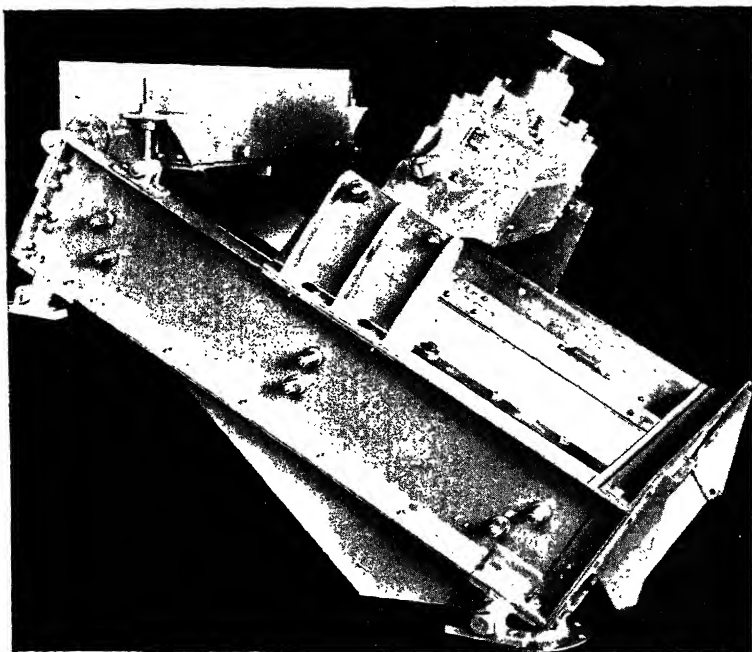


FIG. 10.—Hum-mer electric screen.

rinsed for recovery of the medium for reuse. This draining and rinsing operation is done on horizontal mechanical screens of the reciprocating or circle-throw types.

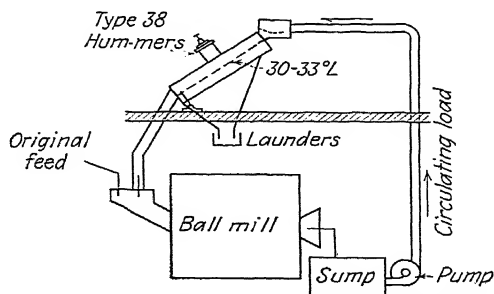


FIG. 11.—Vibrating screen in closed-circuit grinding.

Electric screens equipped with 65- to 100-mesh cloth are employed to clean the reclaimed medium when it becomes contaminated with "fines."

TABLE 3.—STANDARD SCREEN SCALE SIEVES

Tyler Stand- ard Screen Scale $\sqrt{2}$ or 1.414 openings in inches	For closer sizing sieves from 0.0015 to 3.000 in. ratio 4 $\sqrt{2}$ or 1.189	Mesh	Diameter of wire decimal of an inch	U. S. series equivalents (fine series)		British standard equivalent mesh
				Micron designa- tion	Number	
(1)	(2)	(3)	(4)	(5)	(6)	(7)
1.050	1.050	0.148			
.....	0.883	0.135			
0.742	0.742	0.135			
.....	0.624	0.120			
0.525	0.525	0.105			
.....	0.441	0.105			
0.371	0.371	0.092			
.....	0.312	2½	0.088			
0.263	0.263	3	0.070			
.....	0.221	3½	0.065	5660	3½	
0.185	0.185	4	0.065	4760	4	
.....	0.156	5	0.044	4000	5	
0.131	0.131	6	0.036	3360	6	5
.....	0.110	7	0.0328	2830	7	6
0.093	0.093	8	0.032	2380	8	7
.....	0.078	9	0.033	2000	10	8
0.065	0.065	10	0.035	1680	12	10
.....	0.055	12	0.028	1410	14	12
0.046	0.046	14	0.025	1190	16	14
.....	0.0390	16	0.0235	1000	18	16
0.0328	0.0328	20	0.0172	840	20	18
.....	0.0276	24	0.0141	710	25	22
0.0232	0.0232	28	0.0125	590	30	25
.....	0.0195	32	0.0118	500	35	30
0.0164	0.0164	35	0.0122	420	40	36
.....	0.0138	42	0.0100	350	45	44
0.0116	0.0116	48	0.0092	297	50	52
.....	0.0097	60	0.0070	250	60	60
0.0082	0.0082	65	0.0072	210	70	72
.....	0.0069	80	0.0056	177	80	85
0.0058	0.0058	100	0.0042	149	100	100
.....	0.0049	115	0.0038	125	120	120
0.0041	0.0041	150	0.0026	105	140	150
.....	0.0035	170	0.0024	88	170	176
0.0029	0.0029	200	0.0021	74	200	200
.....	0.0024	250	0.0016	62	230	240
0.0021	0.0021	279	0.0016	53	270	
.....	0.0017	325	0.0014	44	325	
0.0015	0.0015	400	0.001	37	400	

Screen Testing.—Testing sieves are used to control crushing and screening operations and to determine the conformity of a crushed or screened product to a given size

series in most general use is known as the Tyler Standard Screen Scale, shown in Table 3. This scale has as its base an opening of 0.0029 in., which is the opening of the standard 200-mesh testing sieve, with the openings increasing in the ratio of the square root of 2, or 1.414. Where a closer sizing is desired, the Tyler fourth root of 2 series, shown in column (2), is used.

The U.S. Bureau of Standards Sieve Series can be used interchangeably with the Tyler Series, the equivalent numbers and micron designations being shown in columns (5) and (6). The British Engineering Standards Association sieves, in column (7), follow closely the openings in the Tyler Series, with some variations in mesh designation to make use of standard British wire gauges.

CHAPTER VI

CLASSIFICATION AND CONCENTRATION

PART I. CLASSIFICATION

BY JOHN V. N. DORR¹ AND FRANK L. BOSQUI²

Definition.—Classification is the art of sorting particles of different sizes and specific gravities into uniform products. As commonly applied in ore dressing, it represents a separation of the coarser sand from the fine sands or slimes in a given ore pulp. Classification is not synonymous with sizing, as for instance by screens, because the latter term does not take into consideration the specific gravity of the particles, which in classification becomes fully as important as the particle size.

Classification is dependent upon the settling rate of the individual particles in a fluid medium (usually water), and the difference in rates between particles of different sizes and/or specific gravities is the controlling factor in the operation.

Settling of particles in water can be divided into two distinct classes—free and hindered settling—and it is necessary to have a clear conception of the two terms to understand the underlying principles of the various machines.

Free settling is said to take place when the individual particles settle freely and unhindered by other particles through a medium of still water or against a rising current of water.

Hindered settling occurs when particles of mixed sizes, gravities, and shapes in a crowded mass are sorted in a rising current of water, the velocity of which is less than the settling rate of the particles, but sufficient to keep the particles in a turbulent or fluid condition. The arrangement of the particles is positive, and if one particle is moved either upward or downward it will return to its original position as soon as set free.

Settlers or thickeners, involving the separation of water from solids by sedimentation, are not treated in this text as classifiers, but are covered in the chapter on Dewatering.

History.—The role of classification in metallurgy is not an innovation of recent years, although the development of the machines in use at this time can be credited to this century. Nature has provided the flowing brook, classifying silt and sand, and it is not unlikely that man's invention of the classifier bears an analogy to the brook. Agricola in Book VIII of his "*De re metallica*" describes the process of removing mud or slime from gold and tin ores by means of sluice boxes in accordance with the metallurgical processes in use in the sixteenth century, and it is probable that what are today termed sand-slime separations have been practiced in ore-dressing plants from the inception of the art of metallurgy.

The surface current or box classifier, later developed by Rittinger with his spitzkasten, was probably the first type used in mills of the present scientific age. In 1868, the hog-trough classifier was installed in the Lake Superior copper mills, and though not satisfactory, was probably the forerunner of the hydraulic classifiers. In

¹President, The Dorr Co., New York.

²Metallurgical Engineer, Westport Plant, The Dorr Co., New York.

Acknowledgment is here made of the assistance of the staff of the Dorr Company, New York.

1881, the Calumet or Richards-Coggin classifier was introduced, the first positive acting machine to be used. Shortly after the Calumet there followed the Evans, Anaconda, Tamarack, Yeatman, and others too numerous to mention, all based upon essentially the same principles but differing in construction.

The hindered-settling principle was recognized by Prof. R. H. Richards as presenting unusually attractive features in the preparation of material prior to concentration, and he and others developed several types of hindered-settling classifiers that are in use in several mills today.

In 1904, J. V. N. Dorr developed his first mechanical classifier at the mill of Lundberg, Dorr & Wilson at Terry, S.D., and the machine quickly obtained recognition. It is today the most widely used type. The invention of the Dorr bowl classifier in 1916 permitted clean separations to be made at as fine as 350-mesh.

Other mechanical classifiers have been introduced, noteworthy among which is the Akins. The Federal-Esperanza and other types of homemade drag classifiers have not proved to be especially satisfactory. They are cheaply constructed but are expensive to operate.

The most widely used of the cone classifiers are the Caldecott and the Allen cones. The former was practically accepted as standard in the older South African cyanide plants, those of the Central Rand, and the latter found considerable acceptance in the Southwest and in the phosphate-rock mining fields. Mechanical-type classifiers have largely replaced cones in recent years.

Types of Classifiers.—Classifiers themselves can be divided into two general classes, hydraulic and nonhydraulic, depending on whether an upward current of "hydraulic" water is or is not used as a sorting column. Nonhydraulic classifiers can in turn be subdivided into mechanical and nonmechanical types and the hydraulic into the free-settling and hindered-settling types. The following outline illustrates this and gives the types described under each class:

I. Nonhydraulic Classifiers

Part A. Spigot type.

1. Spitzkasten.
2. Callow cone.
3. Caldecott cone.
4. Allen cone.

Part B. Mechanical.

1. Dorr classifier.
2. Dorr hydroseparator.
3. Akins classifier.
4. Hardinge countercurrent classifier.
5. Denver hydroclassifier.
6. Drag-belt classifier.
7. Federal-Esperanza classifier.

II. Hydraulic Classifiers

Part A. Free settling—elutriator.

1. Evans shallow-pocket classifier.
2. Richards Launder-Vortex type.

Part B. Hindered settling.

1. Dorreo sizer.
2. Deister (Concenco).
3. Richards.
4. Richards-Janney.
5. Hydrotator (coal cleaning).

The nonhydraulic spigot-type classifier is used nowadays only for rough sand-slime separations, whereas the mechanical type is used almost universally for closed-circuit grinding work. Hydraulic classifiers find their greatest use in the distribution of sized feed to tables and other concentrating machines, and in the production of sized finished products such as in the abrasive and glass-sand industries. The hindered-settling type has practically replaced the free-settling type because of the lower water consumption and high efficiency attained.

Theory.—The theory of classification has been thoroughly investigated by Richards, Rittinger, and others, and only an attempt to summarize their work will be given in this text. A particle suspended in water falls by reason of the unbalanced force represented by the excess of its weight over the weight of water displaced. The law of equal falling particles states that bodies falling freely in water do so at a speed proportional to their weight divided by the resistance of the water. Particles that attain the same final velocity and therefore reach the bottom of a tank at the same time and are found in the same layer are known as equal falling particles.

Free settling of a particle suspended in water is dependent on the following, other conditions being equal in each case:

1. Specific gravity—of two particles of different specific gravities, the one having the higher will settle faster.

2. Size—of two particles of different size, the larger will settle faster than the smaller.

3. Shape—of two particles of the same diameter, the roundish grain will settle faster than the long narrow grain, and the latter will settle faster than the flat grain.

4. Air bubbles—a particle that does not retain adhering air bubbles settles faster than one that does.

5. Magnetism—of two groups of particles, one of which is magnetic while the other is not, the magnetic group will probably settle more rapidly than the nonmagnetic owing to the mutual attraction of the magnetic particles resulting in the formation of flocculated aggregates.

6. Density of liquid—in two liquids of different densities, the rate of settlement is more rapid in the lighter liquid.

7. Viscosity—in two liquids of different viscosities, the rate of settlement of a particle is more rapid in the more fluid liquid.

Opposed to the fall of a particle is the mechanical resistance of the water. This resistance is of two kinds. When the velocity of fall is relatively great, as with large particles, the particle must remove the water from its path, impressing on the water velocity and movement that results in eddies—this resistance is termed “eddy resistance.” When the rate of settlement of the particle is relatively slow, as it is with small particles, the eddy resistance is negligible, the controlling resistance being due to the viscosity of the water or skin friction with the particle—this is termed “viscous resistance.”

Up to a size of about 200 mesh with quartz the viscous resistance predominates with the effect of the eddy resistance increasing as one goes to larger particles and finally predominating over the viscous effects at 10 mesh or so.

For spherical bodies, the general laws of resistance state that in the region of viscous resistance

$$R = 3\pi\mu V \quad \text{Stokes}^1$$

while in the region of turbulent resistance

$$R = K\rho^2 V^2 \quad \text{Newton}^1$$

¹ STOKES, “Mathematical and Physical Papers,” 1901; and *Trans. Cambridge Phil. Soc.*, Vol. 9, Part II, p. 51, 1851.

² NEWTON, “Mathematical Principles of Natural Philosophy,” Book II, trans. into English 1729.

where R is the resistance, d the diameter of the sphere, μ the coefficient of viscosity, and ρ the density of the medium; V is the velocity of the sphere relative to the medium and K is a constant.

Though much published data on the rate of fall of spheres of different kinds in various media are available, Castleman has pointed out¹ that the limits of the laws are not well defined and no simple mathematical expression is possible.

However, by dimensionless reasoning it is possible to show that the resistance S is a function of a factor known as the Reynolds number, which is in itself dependent

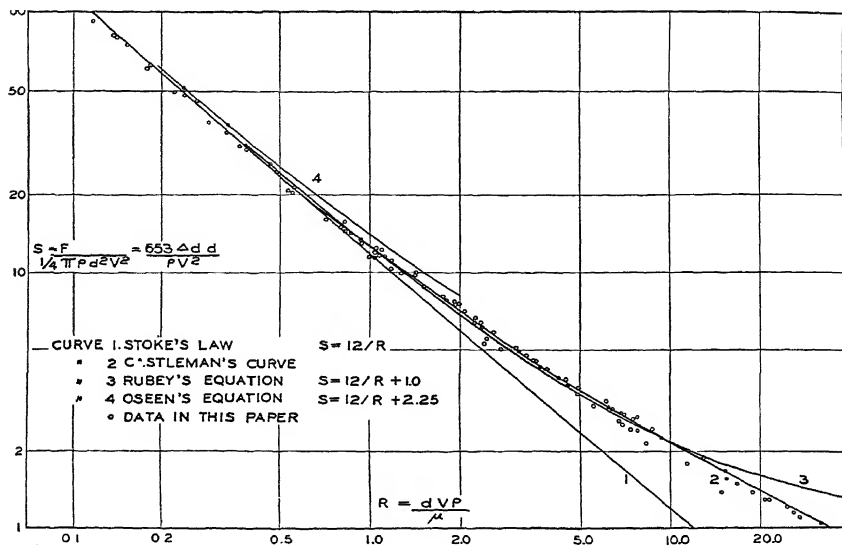


Fig. 1.—Castleman's curve and earlier equations.

on the diameter and velocity of the particles and on the density and viscosity of the medium. A plot of this curve is shown in Fig. 1.

At low velocities, the resistance R of Stokes' equation above can be shown to equal the downward force of gravity and the velocity of the particle expressed as

$$V = \frac{1}{18} g \frac{(\sigma - \rho)d^2}{\mu}$$

where σ = density of the sphere.

g = acceleration due to gravity.

which is the familiar form of Stokes' law.

It can be seen on Fig. 1 that this law holds reasonably well up to about $R_e = 0.5$, above which point inertia forces attain sufficient magnitude to cause considerable deviations. Ritinger and others have derived formulas to cover these higher regions of turbulent flow, but all are approximate or limited in extent.

¹ CASTLEMAN, The Resistance to the Steady Motion of Small Spheres in Fluids, National Advisory Committee for Aeronautics, Tech. Note 231.

Settling Ratios.—Spheres of different substances are equal settling if they bear to each other the proper size ratio. This is called the “free-settling ratio” and is conventionally expressed in terms of quartz. The free-settling ratio of galena is 4.01. This would be the same as saying that a grain of galena of 4.01 mm. diameter would settle at the same rate as a quartz grain of 16.08 mm. diameter. If screens of this size ratio were employed to effect a grading, all the grains of galena could theoretically be separated from all the grains of quartz by jigging.

Where there is a mass of particles of ore in suspension in a fluid such as water, the combined ore and water must be considered as a “fluid” and the specific gravity of this “fluid” must be used in making computations. This favors the settlement and separation of heavy grains smaller than those indicated by the free-settling ratios. It will be evident further that since the heavier grains are hindered by their neighbors

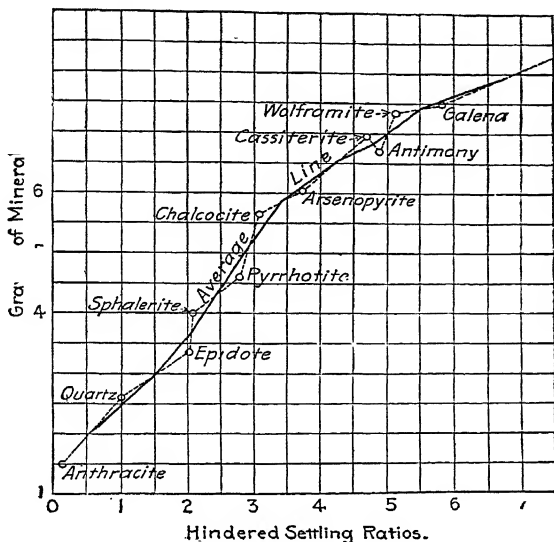


FIG. 2.—Hindered settling ratios for common minerals.

in taking a direct path downward, their average velocity is very much less than their free-settling velocities, and hence the relative settlement rates will be more directly in proportion to specific gravity. Owing to this and other factors, Richards, by experimental work, has determined that these “hindered-settling” ratios are higher than the free-settling ratios. He has determined, for example, that the hindered-settling ratio for galena to quartz is 5.842. The accompanying table shows the free- and hindered-settling ratios for common minerals; the hindered-settling ratio is plotted in Fig. 2 against the specific gravity of the mineral. If two heavy materials are to be separated from one another as well as from quartz, then the hindered-settling ratio of the lighter of the two heavy minerals must be divided into that of the heavier to find the screen ratio, if the hindered-settling ratio of the quartz and the lighter of the two heavy minerals is less than this ratio. For example, the working screen ratio for a mixture of chalcocite, galena, and quartz is not 5.842 or 3.115 but 1.875, the ratio of the hindered-settling factors of the two heavy minerals.

SETTLING RATIOS TO QUARTZ

	Free settling ¹	Hindered settling ²
Sphalerite.....	1.85	2.127
Pyrrhotite.....	2.14	2.808
Chalcocite.....	2.64	3.115
Arsenopyrite..	2.82	3.737
Cassiterite.....	3.32	4.698
Wolframite....	3.64	5.155
Galena.....	4.01	5.842
Native copper.	4.56	8.598

¹ After Rittinger.² After Richards.

It will be seen from the foregoing that spheres of specific gravities S and S_1 are equal settling if

$$(S - \rho)d^2 = (S_1 - \rho)d_1^2 \text{ in the region of Stokes' law}$$

$$(S - \rho)d = (S_1 - \rho)d_1 \text{ in the region of Newton's law}$$

And more generally (where the medium is water)

$$d(S - 1)^n = d_1(S_1 - 1)^n$$

As can be seen from the above, n varies from a numerical value of $\frac{1}{2}$ for the region of Stokes' law to 1 for Newton's law. Since, however, we are concerned in hydraulic classification with relatively fast-settling particles, the higher value may be assumed.

In the case of quartz and galena with specific gravities of 2.65 for quartz and 7.5 for galena, the relation between the diameters would be $(7.5 - 1)/(2.65 - 1)$, or approximately 4. As the volume varies as d^3 , the ratio between the volumes would be 64. With a unit weight of 7.5 for galena and 64×2.65 , or 169.9, for quartz, the relation between the weights would be 23. It is evident therefore that of two equal falling particles the denser particle occupies a much smaller volume as well as having a much smaller absolute weight.

If, instead of using water, a medium of higher density is employed, the ratio between diameters will accordingly be increased. For this purpose a pulp having a density of about 1.5 is usually used. The ratios can then be expressed as follows for quartz and galena:

$$\frac{7.5 - 1.5}{2.65 - 1.5} \text{ or } 5.2$$

The ratio between the volumes under these conditions will be 140 and between the weights 50 instead of 64 and 23 as shown above for free settling.

It is evident that with the higher ratio obtained in hindered settling a separation of two minerals of different densities is more readily accomplished than by free settling, although the actual sizing is less perfect.

This has been confirmed by experiment by Richards. The character of the work done by both free- and hindered-settling classifiers is shown in Figs. 3 and 4, the former showing the result of screen sizing the different spigot products of a free-settling classifier. and the latter the same thing from a hindered-settling classifier. The dark

particles are galena and the light quartz, each vertical column being the result of screen sizing the product from a single spigot. It will be noticed that there is a well-defined valley in the case of the hindered-settling products (Fig. 4) between the galena and the quartz, indicating that separation into concentrate and tailing in the subsequent concentrating machine can readily be accomplished with the making of but little middling. On the contrary, Fig. 3 shows no such valley, indicating that in making the separation of a clean concentrate from a clean tailing by free settling a considerable amount of middling is produced, requiring retreatment, or else a lower grade concentrate and high tailing loss will result.

The hindered-settling classifier is essentially a primary concentrating machine rather than a sizing apparatus, and as such possesses distinct advantages in the preparation of feed for concentrating machines. For the classification of feeds of the same material there is little advantage in the use of hindered-settling classifiers. As, in the

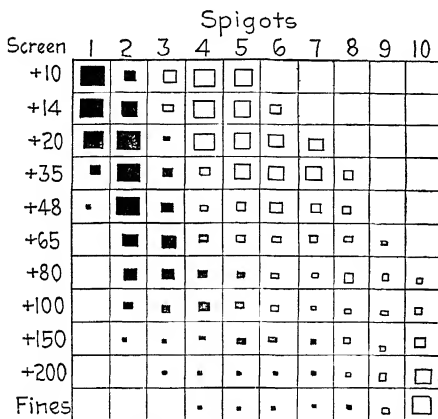


FIG. 3.—Free settling.

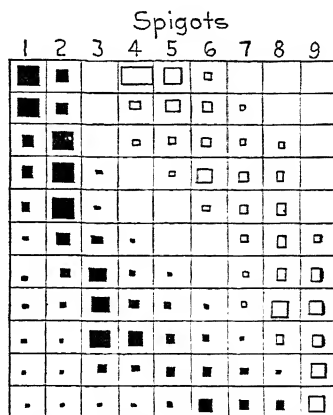


FIG. 4.—Hindered settling.

majority of cases, a classifier working in closed circuit with a grinding mill is treating this type of material, mechanical classifiers are largely employed.

Mechanical classifiers of the Dorr and Akins types are of neither the free-settling nor the hindered-settling types, although they embody some principles of each. The primary factor upon which separations are dependent is the dilution, or ratio of water to solids in the pulp. By varying this, the density of the pulp is varied, and in coarse separations this density may be so high that a hindered-settling condition or jiggling action aided by agitation by the rakes does prevail, but as the dilution is increased to the point where the density approaches that of water a free-settling state exists. But, as in neither case a sorting column with hydraulic water is used, the mechanical classifiers cannot be included in either class. Besides dilution, the rake or helix speed and slope of the classifier tank are functions governing the separation in the machine.

NONHYDRAULIC CLASSIFIERS

A. SPIGOT TYPE

Surface-current Classifiers.—The simplest type of surface classifier is a pyramidal box placed in a stream of pulp, the feed flowing in on one side and overflowing from the other. With the overflow goes the material that does not settle, while the

coarser material settles in the box and is discharged by a spigot or a similar device. This type of apparatus, known as a surface-current or box classifier, is the foundation of the design of the spitzkasten and succeeding types of classifiers (Fig. 5).

If, instead of using one box, a number of them of increasing size are placed in a series so that the pulp flows successively through them, there will be obtained from the boxes a series of graded products, determined by the velocity of the stream in each box and the law of equal falling particles. Such a series of boxes is designated as a spitzkasten series and may consist of any number of boxes.

The horizontal velocity of the pulp is largely absorbed in descending eddies so that it may be assumed that the incoming stream all descends below the surface to rise again and overflow, the material overflowing with it being that possessing a falling velocity less than the velocity of ascent at the overflow. In order to effect a convenient method of discharge, the boxes are made with sides sloping at an angle of not less than 60 deg. To reduce the pressure of water above the thickened pulp and thereby produce a thicker discharge, the discharge pipe is often bent upward forming

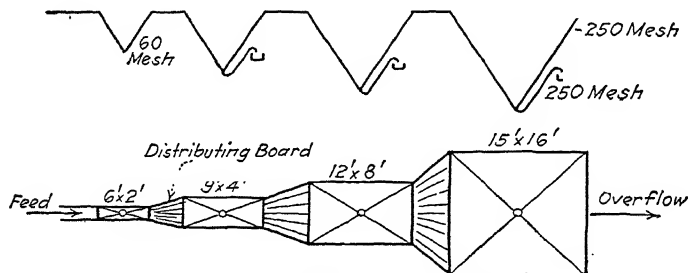


Fig. 5.—Rittinger spitzkasten series.

an inverted syphon, so that the pulp is discharged at a higher level. At the feed side and between the boxes are distributing boards, all being on the same level.

In the Rittinger series, the width of the box increases in the geometric ratio of 2 and the length in the arithmetic ratio of 3. The following tabulation gives the dimensions of a typical Rittinger spitzkasten series, together with the size of products that might be produced.

	Box 1	Box 2	Box 3	Box 4	Overflow
Length.....	6 ft.	9 ft.	12 ft.	15 ft.	
Width.....	2 ft.	4 ft.	8 ft.	16 ft.	
Mesh of product.....	+60	+100	+150	+250	-250

Sometimes, instead of having completely separate boxes, a gradual widening and deepening trough, divided into compartments by vertical partitions, is used. A similar design, in which the intermediate partitions do not reach the level of feed and final overflow, is also occasionally employed, but this last type is more suited to settlement than to classification. Some of the newer types of spitzkasten provide an upward stream of hydraulic water in each box in order to obtain cleaner products.

Although popular at one time, spitzkastens are in little use in America today except in a few small concentrating mills and nonmetallurgical plants, but they are still in

extensive use in European lead and zinc concentrators. The apparatus has the disadvantages of not producing sharply sorted products and necessitating the removal of an excessive amount of water with the spigot products to prevent plugging.

Callow Cone.—This machine is a conical settling tank with a vertical central feed, peripheral overflow, and a spigot in the form of a gooseneck to discharge the settlings. Although occasionally used for making sand-slime separations, the Callow cone is but rarely used as a classifier, its principal function being dewatering, and its use in that capacity will be described in the chapter on that subject.

Caldecott Cone.—In the Rand cyanide plants, the general practice was to use cone classifiers equipped with the Caldecott diaphragm. This type of classifier

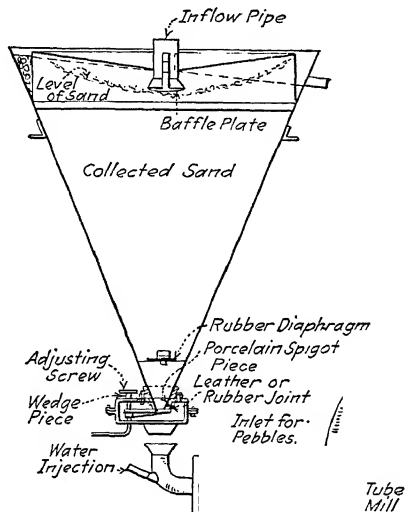


FIG. 6.—Caldecott cone classifier.

(Fig. 6) gave such excellent results that it was practically without competition in that locality, until the advent of the mechanical classifier.

The Caldecott classifier is essentially a cone classifier equipped with a central feed pipe and peripheral overflow launder, the sides of the cone sloping about 20 deg. from the vertical. The discharge is obstructed by a diaphragm placed just above the spigot discharge. The diaphragm is generally a circular plate fixed horizontally to leave between its circumference and the cone a narrow annular passage. Discharge of the settled material is so delayed that the classifier works almost filled with sand, only a basin-shaped depression remaining unfilled. Into this depression the feed falls centrally and sweeps outward toward the rim. The coarse sand settles, while the sufficiently fine material is carried to the overflow. The retardation of the settling of the sand by the diaphragm permits only a regular supply of sand to pass through to the discharge. The formation of vertical channels is minimized, and this fact

combined with the weight of sand above the diaphragm results in a comparatively low moisture content in the discharge.

The customary size of cone is 8 ft. in diameter by 10 ft. in depth. This size cone will deliver 400 to 600 tons of sand per 24 hr. from a pulp at a dilution of 7 or 8 to 1, the feed solids containing about 44 per cent slime and 56 per cent sand, the discharge analyzing less than 30 per cent moisture and usually under 10 per cent slime.

While reasonably efficient and satisfactory for separating sand for regrinding, this type of classifier does not achieve a close sizing, about 10 per cent slime being found in the sand and a similar percentage of coarse material in the overflow.

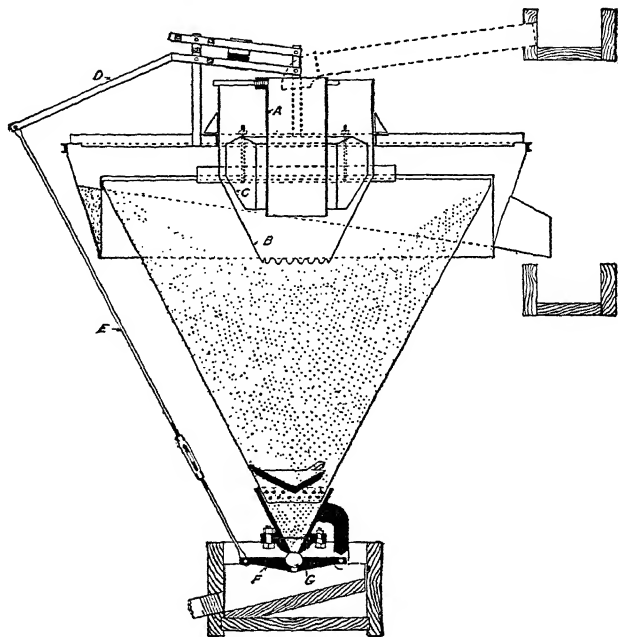


FIG. 7.—Allen cone.

Allen Cone.—The Allen cone (Fig. 7) is similar to the Caldecott in its essential features, the chief distinction being in the automatic discharge. The Allen cone has attained considerable recognition in the southern phosphate-rock washing plants and in the southwestern mining field where its principal function is the separation of a mixed pulp into gravity concentration and flotation feed. It is also used for classifying in a regrinding circuit, for desliming sand, and for dewatering tailings and concentrates.

The feed enters from a launder to a feed spout *A* and, after passing through a truncated cone *B*, the water and solid particles which it is desired to remove flow upward and out into the overflow launder which encircles the cone. The sand particles settle in the cone and form a basin. When the settled sand reaches the outlet of the truncated cone, it obstructs it so that the water rises in *B*, lifting the float *C*. The upward movement of the float depresses the levers *D*, *E*, and *F*, so that the ball

valve *G* is dropped from its normal position. The sand is discharged in a continuous stream until the level of the pulp is brought below the mouth of the truncated cone, when the ball rises and closes the opening. The object of the diaphragm is the same as in the Caldecott cone, *viz.*, to prevent sudden rushes of sand through the spigot and to cause upward-moving currents of water so that the slime may be kept in suspension.

A feature of the Allen cone is that its operation is not affected by fluctuations in the amount of feed, the valve opening only when there is sand to be discharged. The size of particles in the overflow is dependent on the dilution of feed.

A typical example of the work of the Allen cone is given in the following tabulation. In this instance, these cones were used to prepare the feed to vanners and flotation at the Old Dominion Copper Concentrator.

Feed to three cones.....	80 dry tons in 24 hr.
Moisture in feed.....	89.4 per cent
Moisture in spigot discharge.....	28.6 per cent
Moisture in overflow.....	94.4 per cent

B. MECHANICAL TYPE

The **Dorr classifier** (Fig. 8) is essentially a rectangular settling tank in the form of an inclined trough, open at the upper end for the discharge of the oversize particles and provided with an overflow dam across the closed lower end, where fine particles and slime are overflowed. The inclination varies from $1\frac{1}{2}$ in. per ft. upward, depending upon the mesh of the separation to be made. A reciprocated raking mechanism carries settled solid particles up the inclined deck to the sand discharge lip.

Feed enters near the overflow—through either end of a transverse trough equipped with vanes for providing even distribution across the width of the tank.

The raking mechanism, involving one to four rakes, depending upon the type and size of machine, is swung from hangers, one at the overflow and one at the sand-discharge end. All bearings are above the pulp level. The machine is designated as simplex for one rake, duplex for two rakes, and quadruplex for a machine with four rakes. The head motion, located at the sand-discharge end of the classifier, actuates the raking mechanism through a system of heavy gears, cranks, and eccentrics with the necessary links and hangers. A manual or motorized lifting device is provided to raise the lower end of the rakes.

This head motion is such that the rakes in the forward raking stroke move parallel and close to the inclined deck, while on the return stroke the rakes are lifted through a path in which the upper corners of the motion are rounded off. Just before the start of the raking stroke, the rake blades are dropped vertically into the sand, which they then convey forward. The length of stroke is such that the sand that one rake brings up is caught by the next rake on the succeeding stroke.

The head motion may be driven from a line shaft or may be directly connected to an individual motor drive. A motor and V-belt drive is a preferred arrangement. In cases where operating conditions are irregular, a variable-speed drive is employed to advantage.

With correct dilution and rake speed, a pulp fed to the classifier rapidly segregates into (1) the coarse quick-settling material, larger than the mesh of separation, and (2) the fine slow-settling material, finer than the mesh of separation. The coarse portion sinks to the bottom and is conveyed up the inclined deck by the reciprocating rakes. The fine portion, held in suspension by the specific gravity of the pulp and the degree of agitation of the rakes, is carried to the overflow lip where it is carried out of the classifier.

The motion of the reciprocating rakes tends to release undesired fines from the sand bed through agitation and slight attrition. Water sprays applied above the pulp level further aid in removal of fines and entrained solution. Removal of a large portion of accompanying free moisture from the sands is accomplished by compression on the upper end of the inclined drainage deck above the pulp level.

With the proper size and type of classifier, the mesh of separation is controlled by means of various adjustments such as pulp dilution, tank slope, and rake speed. Generally speaking, the greater the percentage of solids in the bath and the higher the rake speed, the coarser will be the mesh of separation.

Dorr classifiers are furnished in four models. Model FX is for heavy tonnage and high circulating load work, model F for average conditions, model FH for intermediate

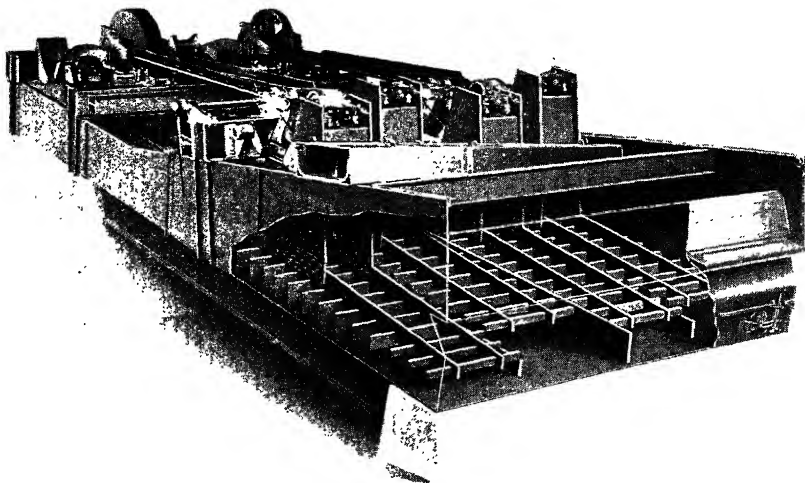


FIG. 8.—Dorr quadruplex classifier.

duty, and model FR for light duty. The principles of operation in all four are identical, but the details of construction and mechanical features differ, as do the fields of application of the classifiers themselves. Figure 8 shows a quadruplex model FX classifier.

Dorr FX Classifier.—This is a rugged, heavy-duty machine, weighing twice as much as a type F unit of the same width and containing extra-heavy-gauge steel tank and cast- or welded-steel mechanism parts. Because of the deeper rake blades and longer stroke, its sand-raking capacity is $2\frac{1}{2}$ to 3 times that of the type F machine. It has pressure-lubricated antifriction bearings.

Dorr F Classifier.—Where the duty is not such as to justify the type FX machine, the type F unit is recommended. It is similar in general appearance to the heavy-duty FX, but its construction is in line with the relatively less severe duty of average practice.

Dorr FH Classifier.—This intermediate-duty classifier is used in service ranging between that requiring the type F and the light-duty FR. It is suitable for low-tonnage closed-circuit grinding work.

Dorr FR Classifier.—The FR is the lightest in the Dorr classifier line and is used for light duty in relatively small plants. It is particularly suitable where the rake-product tonnage is light compared with the overflow.

SIZES AND TYPES OF DORR CLASSIFIERS

Type	Range of widths	Range of lengths
FX—heavy duty.....	5' 0" to 16' 0"	24' 0" to 31' 6"
F—normal duty.....	6' 0" to 16' 0"	18' 4" to 30' 0"
FH—intermediate duty.....	4' 0" to 5' 0"	18' 4" to 30' 0"
FR—light duty.....	1' 6" to 4' 0"	12' 0" to 23' 4"

Dorr Bowl Classifier.—The Dorr bowl classifier (Fig. 9) is used for obtaining a very fine overflow, where overflow capacity requirements are greater than would be pro-

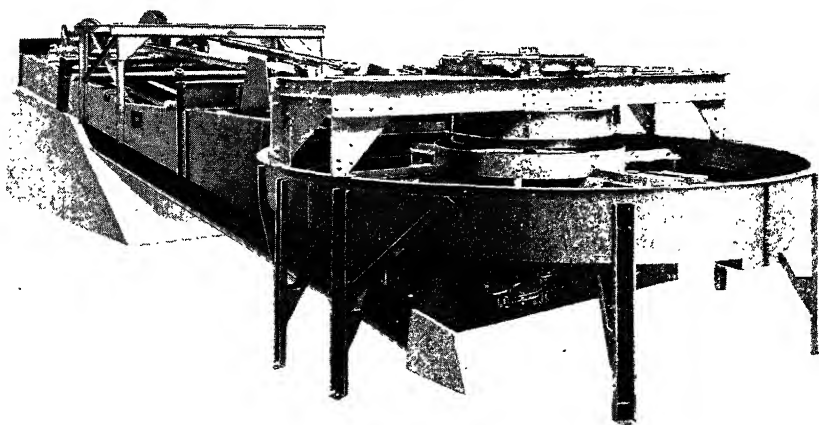


FIG. 9.—Dorr bowl classifier, duplex rakes.

vided by a straight-line classifier, or where extremely clean sands are desired. It consists of a shallow steel bowl with a rotating raking mechanism, similar to that of a thickener, mounted on the lower end of a Dorr FX, F, FH, or FR classifier, depending upon the sand-raking requirements.

The bowl has a conical bottom sloping slightly toward the center. The feed enters the bowl through a shallow feed well and over a distributor plate at the center. The fine light suspended solids overflow into a peripheral overflow launder. The coarse heavy solids that settle in the bowl are carried by the revolving rakes to an adjustable central-discharge opening where they drop into the reciprocating rake compartment below.

The bowl classifier is a two-stage classifier, in which the primary classification takes place in the bowl, producing a finished overflow and a sand product discharged as a semisolid to the reciprocating rake compartment. The secondary classification takes place in the rake compartment where the sands are turned over by the rakes and the slime and fine sand are brought into suspension and returned to the bowl.

The action of the plows and the rakes, together with the counterflow of wash water, will produce very complete washing and separation. Great flexibility is attained since, in addition to the adjustments of rake speed and dilution, the speed of the bowl rakes and the distribution of wash water can be varied.

Because of greater classification area, separations may be made at higher pulp densities than in the single-stage classifier.

The critical size control is a feature of the Dorr bowl classifier that provides a positive recirculation of critical mesh material from the reciprocating rake compartment beneath the bowl back into the bowl itself. Its use increases the over-all classification efficiency.

Bowl diameters may range from 3 to 28 ft., depending upon the separation requirements. They may be mounted on any model or size of reciprocating rake mechanism, except of course that the bowl diameter must be greater than the width of reciprocating rakes.

Dorr Hydroseparator.—The Dorr hydroseparator (Fig. 10) is used for making extremely fine separations, generally in the range of 325 mesh down to the micron-

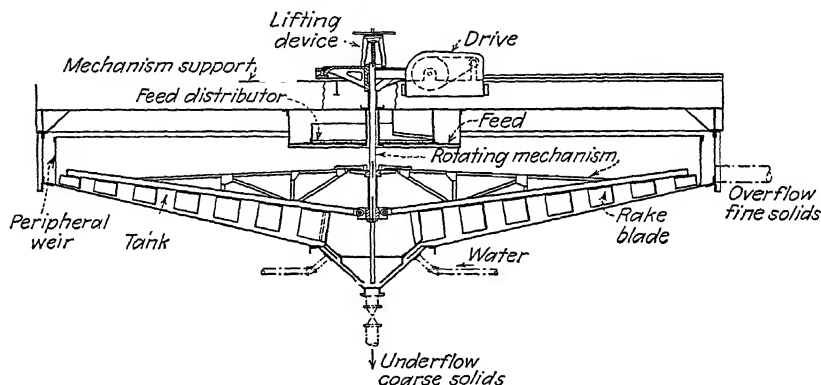


Fig. 10.—Dorr hydroseparator.

size range. Such operations usually require a large classification area per unit of capacity and a considerable pulp depth to minimize the agitating effect of the revolving mechanism and introduction of feed. Coarser separations are sometimes made in the hydroseparator in open-circuit work where large volumes of slime and water are handled.

The hydroseparator tank should have just sufficient area to allow the particles coarser than the mesh of separation to settle out, but insufficient to allow settlement of finer particles. Feed enters through a semisubmerged central feed well with the suspended material overflowing into a peripheral launder. The solids that settle are carried to the center by the revolving raking mechanism and removed through a washing-type discharge cone in the center of the bottom of the tank.

Operating Conditions.—A Dorr rake classifier may be adjusted to deliver a product as coarse as 14 mesh to 200 mesh and finer. By increasing the slope of the classifier tank and the speed of the rakes, a coarser separation is made because the fine particles will run back more readily on a steep slope than on a flat slope, and because the greater mechanical agitation at the higher rake speeds will hold coarser particles in suspension and cause them to overflow.

Pulp dilution, also, has an important bearing on size of separation. The more water there is present, within certain limits, the finer will be the separation made. This is true up to the dilution point where added velocity outweighs dilution effect. Separations in the range between 14 and 65 mesh are materially affected by dilution changes; in the range of fine separations, dilution changes have a less pronounced effect.

The capacity of a classifier is greater with coarse meshes of separation than for fine separations. It is a function of rake speed, tank slope, and percentage of solids in the overflow. High rake speeds and a high percentage of solids are used for coarse separations in the 14- to 65-mesh range, while the opposite is true for finer separations in the 65- to 200-mesh range. Other factors remaining the same, overflow and sand-raking capacity are functions of the width of a classifier tank.

In Table 1 are given some typical conditions of capacity, mesh of separation, rake speed, tank slope, and percentage of solids in the overflow for an average-duty type F classifier on 2.7 specific gravity ore. Capacities are given in net tons of dry solids per foot width per 24 hr. A rough approximation of the capacity of a Dorr type F classifier can be obtained by multiplying these figures by the classifier width. Also given with this table are correction factors that may be used in the case of FX, FH, or FR machines. While this table may be used as a guide for rough preliminary estimating purposes, it is advisable to consult the classifier manufacturer for his recommendation on the proper type and size of machine for any specific job.

Classifier Capacity.—In the case of bowl classifiers, the greater the tonnage and volume to be overflowed and the finer the separation, the greater must be the bowl area. Sufficient reciprocating rake width is essential for proper bowl operation, and therefore capacities per foot of rake width are usually lower than in the case of single-stage classifiers. Proper bowl diameter, width and length of reciprocating rakes, overflow dilution, and rake speed should be recommended by the classifier manufacturer.

TABLE 1.—CAPACITIES¹—STANDARD DUTY DORR F CLASSIFIER
Capacity per Foot of Tank Width—All Widths in Proportion

Mesh of separation	Tons of dry solids per 24 hr. per ft. of width		Rake strokes per minute	Slope, in. per ft.	Per cent solids in overflow
	Overflow capacity	Sand raking capacity			
28	130-170	320-450	23-32	2 $\frac{3}{4}$ -3 $\frac{1}{4}$	33
35	105-145	300-400	21-29	2 $\frac{3}{4}$ -3	25
48	85-115	260-360	19-26	2 $\frac{1}{2}$ -3	22
65	70- 90	235-330	17-24	2 $\frac{1}{2}$ -2 $\frac{3}{4}$	18
100	50- 70	210-300	15-21	2 $\frac{1}{4}$ -2 $\frac{1}{2}$	14-17
150	35- 50	150-235	11-17	2 -2 $\frac{1}{4}$	9-14
200	20- 30	125-200	9-14	2 -2 $\frac{1}{4}$	6-10

¹ Average conditions for 2.70 specific gravity ore.

Power Consumption.—For preliminary estimating purposes it may be figured that average power consumption for Dorr rake classifiers is $\frac{1}{2}$ to 1 hp. per ft. of rake width, plus 50 per cent to take care of momentary requirements during starting up. The power, of course, varies with the sand load to be raked. For driving the bowl, it may be figured that this will vary from $\frac{1}{4}$ hp. for the smallest bowls up to 5 hp. for the largest size.

The "low-weir" type was the original standard Akins design, with the overflow weir below the lower shaft bearing. This gave limited pool area and overflow capacity and is out of date.

The "high-weir" type has the overflow weir above the spiral shaft but still below the top of the spiral. The weir height is adjustable, with a corresponding variation in pool area. This type is used, usually, for separations coarser than 100 mesh. The high-weir type is shown in Fig. 11.

The submerged-spiral type has a pulp bath of greater depth and area, with the lower end of the spiral completely submerged. In this type the pool surface is undis-

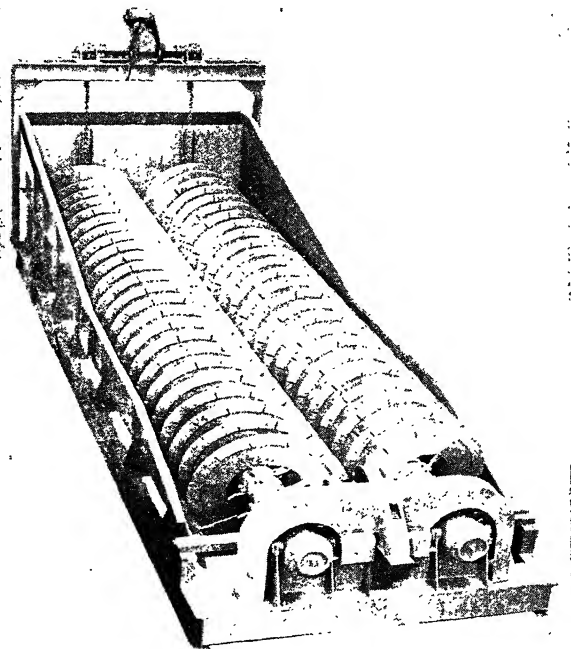


FIG. 12.—Akins duplex submerged spiral classifier. (Courtesy of Colorado Iron Works.)

turbed. This is normally employed for the making of finer separations. The submerged-spiral design is shown in Fig. 12.

All sizes in both the high-weir and submerged-spiral types are furnished in both simplex and duplex design. The spiral diameters available in the high-weir type vary from 16 to 78 in. In the submerged-spiral type the standard sizes vary from 16 to 48 in.

In the Akins classifier, the tank slope, speed of rotation of spiral, dilution of pulp, and feed rate control the mesh of separation. The slope is usually from $2\frac{1}{2}$ to $3\frac{1}{2}$ in. per ft. Wash sprays can be used above the pulp level to assist in the removal of entrained slimes from the sand load.

The uses of the Akins classifier are similar to those of the Dorr, the large field being in closed-circuit grinding work. Operating costs, involving power and maintenance, are low.

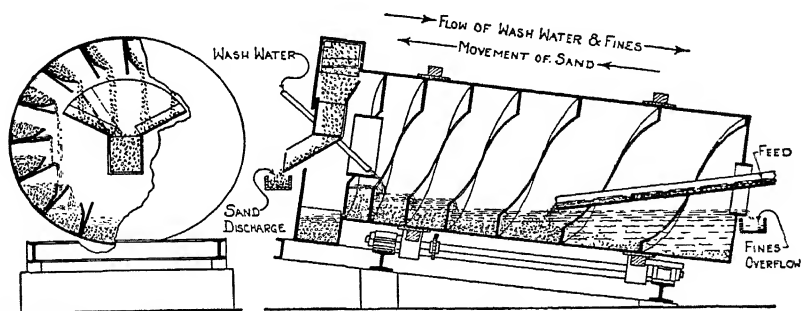


FIG. 13.—Hardinge counter-current classifier.

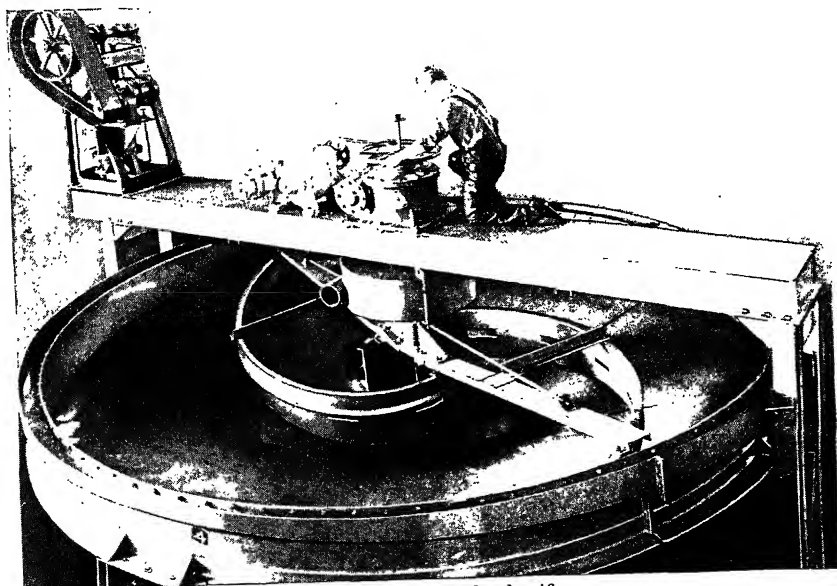


FIG. 14.—Denver hydroclassifier.

Hardinge Countercurrent Classifier.—This classifier consists of a welded-steel drum rotating around an axis at a slight slope from the horizontal, and mounted on tires and rollers. Inside the surface of the drum is a rigidly attached continuous spiral arranged so that, as the drum rotates, a settled sand bed is carried up the slope at which the drum is mounted and discharged at an elevation above the pulp bath depth.

Figure 13 is a sketch of the arrangement of the Hardinge classifier. The feed is introduced normally through the lower end of the drum, and the fine sand and slime are removed over an overflow weir which maintains the pulp pool. Settled coarse solids are moved out of the pool against a countercurrent flow of wash water. Lifting buckets elevate the sand product for return to the grinding mill when the classifier is used for closed-circuit work.

Either standard or extra-length classifiers are available. Control of separation is effected through the speed of rotation or drum, pulp dilution, and depth of pulp pool. Sizes available are from 18 in. up to 10 ft. in diameter. The length of the drum varies from 4 to 40 ft.

The Denver hydroclassifier (Fig. 14) consists of a shallow thickener bowl with a hydraulic washing unit added at the central discharge point, for additional washing

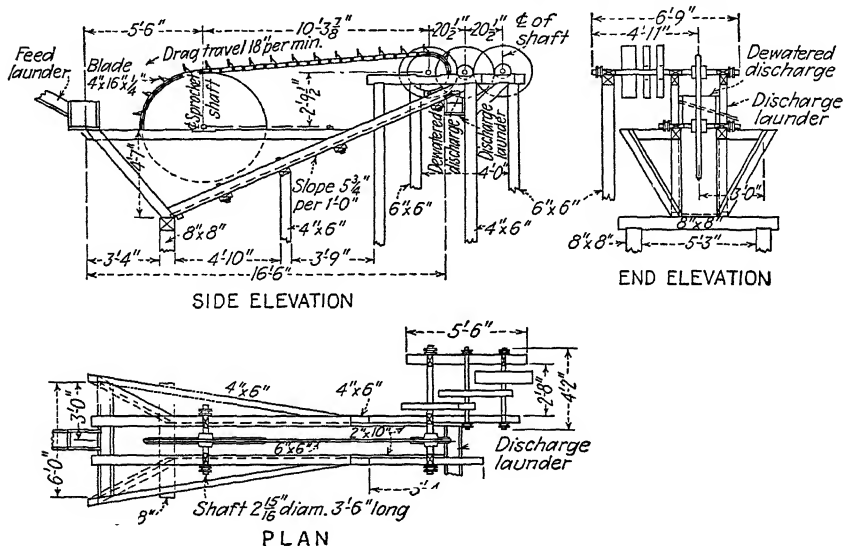


FIG. 15.—Federal-Esperanza classifier.

of underflow solids in hydroseparation. It is built in sizes 8 ft. in diameter, with 25 tons daily capacity, to 30 ft. in diameter with 750 tons daily capacity. The hydroclassifier is used principally for fine sizing and desliming.

Drag-belt or drag-flight classifiers were rather widely used many years ago but have been largely supplanted by modern mechanical classifiers. Drag-type classifiers are usually homemade affairs and vary widely in design and performance. They consist simply of a continuous belt or link chain moving over head pulleys or sprockets so located that part of the travel of the belt is below and part above the level of pulp in a tank. The belt or chain is usually equipped with flights for conveying the solids that settle in the tank. An overflow weir allows removal of suspended fine material, slime, and water. Sprays are sometimes used on the deck or belt, above the pool level, to aid in slime removal from the sand product.

The Federal-Esperanza, shown in Fig. 15, is typical of one of these classifiers. It consists of an inclined settling tank and has a chain, carrying iron blades about 4 in.

high and 16 in. wide, moving at a speed of 1.5 ft. per min. over sprocket wheels set 10 ft. apart. The lower sprocket wheel is large enough so that its bearing may be above the tank pulp level. The feed enters near the lower end, the slime overflows at the same end, and the sand is dragged by the flights up the sloping deck, out of the bath, and is discharged at the upper end. A serious disadvantage of this classifier is the excessive wear of chain pins and bushings when handling gritty material.

Free-settling Hydraulic Classifiers.—The basic principle of the free-settling hydraulic classifier is to utilize the varying settling rates of particles of different sizes or specific gravities to obtain a series of graded products. The sands, while being moved forward by the carrying current, are subjected to a series of upward hydraulic currents, usually dropping in velocity through a series of pockets. The grains that are heavy enough to settle against the rising current in the first pocket do so and are withdrawn through an orifice or spigot in the bottom, while the lighter grains pass

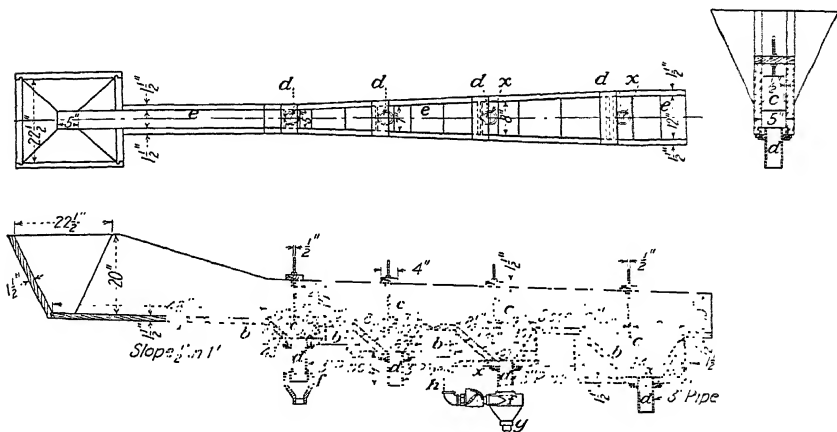


FIG. 16.—Richards launder-vortex classifier.

on to the pockets with successively lower hydraulic velocities, where in turn they are collected and withdrawn.

The *Evans classifier* is a simple type of launder classifier consisting of a sloping launder to which are attached a number of "pressure boxes" and baffles. Hydraulic water is introduced through pipes into each of the boxes, with valve control on each separate line.

Richards Launder-vortex Classifier.—The enlarging and deepening of the pockets and the introduction of sorting columns below the pockets were a forward step in obtaining more closely controlled sizing. These features are embodied in the Richards launder-type vortex classifier shown in Fig. 16. This classifier consists of a shallow trough with four pockets, similar to the earlier Calumet. Each pocket is fitted with a sorting column that consists of a 3-in. pipe about 9 in. long with a vortex fitting at the bottom. The hydraulic water entering this vortex receives a whirling motion, and its helical path in the sorting column prevents the formation of stray downward and upward currents which carry light grains into the spigot and heavy ones into the overflow. Fins at the top of the sorting column stop the whirling motion, and adjustable gates above the pocket regulate the flow. This classifier will handle about

60 tons per 24 hr. of dry material, consisting of about 75 per cent quartz and 25 per cent pyrite, that has passed a $\frac{3}{16}$ -in. round hole and forms a pulp of 10 parts water to 1 of solid.

In the "Textbook of Ore Dressing," third edition, by Richards and Locke, pages 152-153, other types of free-settling classification devices are illustrated and described. Material will be found there, also, concerning methods of calculating the design requirements for classifiers of this type.

Hindered-settling Classification.—The theory of hindered-settling classification has been covered in a previous part of this chapter. Hindered settling is described by Gaudin¹ as "taking place where particles of mixed sizes and gravities in a crowded mass, yet free to move among themselves, are sorted in a rising current of water, the velocity of which is less than the free falling velocity of the particles, but yet fast enough so that the particles are in motion."

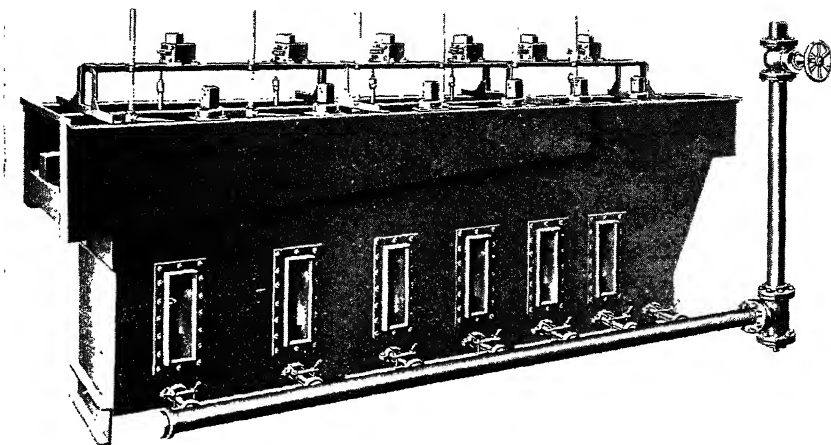


FIG. 17.—Dorcco sizer.

Hindered-settling classifiers are accordingly those wherein fresh water rises through the sorting tube or orifice into a chamber so dimensioned that the material collected there is brought into a "quicksand" suspension or "teeter." The suspension acts as if it were a heavy fluid medium and gives an exaggerated differential effect of settlement based upon differences in specific gravity of mixed particles.

Hindered-settling classifiers are well adapted to the preparation of feeds to concentrating tables, since they produce products in which the coarse light mineral grains are grouped with fine heavy-gravity particles. This type of classification is used for direct concentration of some ores in a relatively coarse size range where there is an appreciable difference in specific gravity between mineral and gangue. Hindered-settling classifiers are used also for separating materials of uniform specific gravity such as silica sand into sharp size bands.

The Dorcco Sizer.—This machine (Fig. 17) consists of a series of compartments, increasing in cross-sectional area from the first to the last. Hydraulic water to each compartment is controlled so that the greatest rising current is maintained in the first

¹ GAUDIN, "Principles of Mineral Dressing."

TABLE 2.—DORRCO SIZER PERFORMANCE ON MINUS 4-MESH LEAN IRON ORE

	Mesh	Per cent weight of original	Per cent weight cum.	Per cent Fe	Cum. per cent Fe grade	Cum. per cent Fe recovery
Heads		100.00	100.00	56.24		
Spigot 1	+ 6	9.29	9.29	61.82		
	+ 8	6.79	16.08	62.27		
	+ 10	3.17	19.25	62.27		
	+ 14	1.84	20.59	64.06		
	- 14	0.64	21.23	57.79		
Total.....	21.23	21.23	62.04	62.04	23.40
Spigot 2	+ 6	3.09	24.32	58.02		
	+ 8	4.93	29.25	59.81		
	+ 10	4.17	33.42	60.48		
	+ 14	3.38	36.80	62.05		
	+ 20	1.96	38.76	63.62		
	- 20	0.97	39.73	59.36		
Total.....	18.50	39.73	60.49	61.30	43.27
Spigot 3	+ 8	1.61	41.34	50.18		
	+ 10	2.11	43.45	54.21		
	+ 14	3.51	46.96	56.22		
	+ 20	4.80	51.76	60.93		
	+ 28	4.59	56.35	64.29		
	- 28	1.63	57.98	63.17		
Total.....	18.25	57.98	59.30	60.69	62.51
Spigot 4	+ 20	2.16	60.14	43.46		
	+ 28	4.85	64.99	56.90		
	+ 35	6.03	71.02	62.72		
	- 35	1.64	72.66	65.73		
Total.....	14.68	72.66	58.31	60.20	77.72
Spigot 5	+ 35	1.71	74.37	35.17		
	+ 48	4.40	78.77	50.85		
	+ 65	4.51	83.28	62.05		
	- 65	1.73	85.01	65.63		
Total.....	12.35	85.01	54.70	59.69	89.75
Spigot 6	+ 35	1.40	86.41	27.10		
	+ 48	2.47	88.88	39.87		
	+ 65	2.16	91.04	53.54		
	+100	1.41	92.45	54.66		
	+150	0.70	93.15	46.14		
	+200	0.70	93.85	33.82		
	-200	0.64	94.49	26.21		
Total.....	9.48	94.49	42.40	57.72	96.89
Slime overflow..	5.51	100.00	31.81	56.29	100.00
Grand total	100.00	100.00	56.29	56.29	100.00

compartment, diminishing to the last pocket. The overflow is taken off through a launder along the sides and the end.

The novelty in this machine lies in the method employed for controlling the density and the amount of material discharged from each pocket, automatically, within close limits. A sensitive hydrostatic pressure instrument, which reacts immediately to changes in pulp gravity, gives positive action in varying the opening in a discharge valve by raising and lowering a valve stem from which a valve plug is suspended.

The control instruments are adjustable for any desired pulp density in order to obtain the desired particle size or mineral grade. When the feed to the sizer is deficient in material of a certain settling rate, the valve in the pocket which collects that particular fraction shuts off until sufficient accumulation takes place to cause that valve to open. When a greater amount of that same fraction occurs in the feed and collects in that pocket, the valve opens wider. The instruments control the point at which the valve begins to open as well as the differential range over which it functions.

Table 2 is typical of many results that have been obtained on minus 4-mesh low-grade iron ore screened from feed to a heavy-medium plant.

Deister "Concenco" Classifier.—This is a hindered-settling classifier consisting of a series of sorting compartments of uniform size. The portion of the tank above the compartment dams is flared outwardly. By regulating the amount of hydraulic water to each compartment, a series of banded products is obtained, grading from coarse at the feed end to fine at the overflow end. It is an improved model of the "St. Joe" classifier, developed originally for sizing table feed in the St. Joseph, Mo., lead district.

The discharge of products from the various compartments is controlled by the use of interchangeable orifices. Product uniformity is dependent on uniformity of feed. Its principal use is in the preparation of table feed, and it is furnished with any number of compartments up to fourteen.

Richards Hindered-settling Classifier.—This classifier, invented by Prof. R. H. Richards, consists of a box of wood or steel having a sloping bottom increasing in width and depth toward the overflow end. Cylindrical cast-iron chambers are bolted to the bottom of the box at intervals, forming the respective spigots. Each chamber consists of two concentric parts, one the outer hydraulic chamber which is tapped for the wash-water pipe; the other, the hindered-settling diaphragm where the final classification takes place, and which forms the necessary constriction. This diaphragm is in the form of a cone pierced by alternating rings of radial and tangential holes, through which the rising water passes from the hydraulic chamber, forming alternating radial and vortex jets within the cone intended to break up any agglomeration of particles and eliminate harmful currents. Water consumption may be 200 to 700 gal. per ton of ore.

Richards-Janney Classifier.—This type of classifier was designed by F. G. Janney and embodies the Richards hindered-settling principle and the Janney features of intermittent discharge, retardation of flow, controlled settling, and hydraulic regulation. The pulp in the sorting column is gently agitated by paddles attached to a hollow shaft, rotated at 80 to 90 r.p.m. by worm gearing on a line shaft. The paddles assist the current of hydraulic water in washing into the next compartment the under-sized particles and in keeping the sand from banking and plugging the sorting column. A ball valve, at the bottom of the sorting chamber, operated by a rod passing through the hollow shaft and a cam on the rotating gear, opens periodically to permit the pulp in the sorting column to drop into a retarding chamber from which it slowly drains through an orifice. The admittance of hydraulic water is controlled by a gate valve and may be closely adjusted to allow particles of sand of any desired size to settle to the bottom of each sorting chamber.

Deister Cone Baffle Classifier.—This (Fig. 18) consists of a cast-iron truncated cone that may be bolted to the bottom of a pointed box, conical settling tank, or launder. The classifier is about 2 ft. long and 8 in. in diameter at the top, tapering to 6 in. at the bottom. The interior of the classifier contains a series of conical diaphragms that are slotted to permit passage of the pulp. The hydraulic water enters through a pipe, passes into the classifier through radial holes in the side, and in rising encounters the descending ore grains. Ample opportunity is afforded the lighter particles to be separated and washed over the top, while the heavier ones settle and are discharged through an orifice in the bottom.

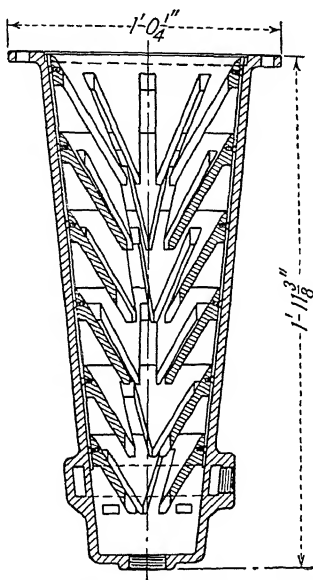


FIG. 18.—Deister cone baffle classifier.

Richards Pulsator Classifier.—The principal feature of the Richards pulsator is the pulsating effect imparted to the hydraulic water, causing it to flow in spurts, keeping the pulp in an intermittent state of agitation.

This classifier consists of three compartments, a settling pocket, below which is a sorting column, and a hutch where the hydraulic water enters. The water is admitted through a pulsator or plug valve, rotated at 120 r.p.m. by a pulley driven from the concentrator shaft. An air chamber is provided to act as a cushion. The water passes upward from the hutch into the sorting column, which is about 9 in. high and $2\frac{1}{2}$ to 6 in. in diameter, depending on the size of the feed and product, and thence to the settling pocket. The supply of water is regulated by a valve, and the classified material falls into the hutch from which it is discharged through a hole in a wooden plug, the size of the hole being determined by the grade of the product.

PART II—MECHANICAL CONCENTRATION

BY DONALD M. LIDDELL¹

Methods.—Mechanical concentration (ore dressing) is the recovery of the valuable portion of an ore through physical differences between the mineral and the gangue, and without resort to chemical changes. Those properties entering into mechanical means of separation are: hardness or softness; tenacity, brittleness, or friability; structure and fracture; aggregation; color and luster; specific gravity and settling power; adhesion; greasiness; electroconductivity; magnetic susceptibility; changes due to heat: (1) decrepitation, (2) changes in porosity and density, (3) changes in magnetic permeability.²

Hardness and tenacity of the ore as a whole are rather more economic factors entering into the costs and hence into the success of the process, than affording means of separation of ore and mineral. Structure and fracture affect the processes of separation to be adopted, as does also the degree of aggregation of the mineral. Color and luster are the properties on which hand sorting depends. Specific gravity and settling power are the foundations of table and jig work. Adhesion is made use of in amalgamation and in diamond recovery (the diamond will adhere strongly to a greasy plate; the gangue but little). Greasiness, the power of attracting air bubbles, was formerly a detriment—now we purposely produce it and thereby float the valuable minerals. Upon heating, some minerals (*e.g.*, calcite, fluorite, and barite) decrepitate strongly. Heating, screening, pulverizing, classifying, and tabling separate such minerals from the gangue. Upon heating, some minerals become magnetic, when they were not so originally, and they can then be separated from a non-magnetic gangue.

The above recital of principles upon which the processes depend is likewise a key to the methods of separation: by differing rates of fall in a surrounding fluid, or along inclined planes, the latter being either with or without a liquid medium; by interstitial effects accompanied by washing; by magnetic or electrostatic methods; by flotation.

The chief types of concentration machines employing water as a medium of separation are the jig, the end-shake table, the round and canvas tables, and the vanner or other side-shake apparatus, the last treating the finest material, the first treating the coarsest.

Crushing for Concentration.—The general subject of crushing and grinding is treated elsewhere in this volume. It must be emphasized here, however, that the object of crushing should be to make a minimum of fines and dust, since this is the material most difficult to treat. Preliminary crushing is principally done in jaw and gyratory crushers, rolls, disk crushers, and stamps. Secondary crushing is done in rolls, stamps, edge runners (Chilean and Huntington mills), and in ball, rod, and tube mills.

Testing for Concentration.—Milling is not merely recovery of a valuable mineral in comparatively pure form from an ore—milling is engineering. High metallurgical recovery is not always high economic efficiency. Ore is only rock unless a profit is made from it. The best mill practice is that which makes the most money from a ton of ore, regardless of the percentage of mineral recovered.

¹ Consulting engineer, New York.

² RICHARDS, "Ore Dressing."

But it cannot be sufficiently emphasized that tailings ought to be impounded. The tailings of one generation are the ore of the next. It does not take a long experience in metallurgy to remember when the tailings in Lake Linden, Mich., were waste—now they are ore—nor to have seen many of the old smelter dumps of the West mined and resmelted.

Limit of Concentration.—With any tailing or middling, tests will disclose the recovery by retreatment, and calculation will show the financial return. And that return must show something well over operating costs and interest and amortization on the machinery and buildings involved in the additional recovery, or the process should not be installed.

Hand Sorting.—The case for hand sorting is stated in one paragraph by S. J. Truscott.¹ "Sorting, which requires only a relatively cheap installation, cheaply removes a substantial amount of material from the ore passing to the most costly mechanical-dressing plant, which may be smaller in consequence. Smaller size of plant also follows from the fact that in removing the extremes of ore and waste, the ore to be mechanically dressed is maintained more regular in value, a condition under which all machines work best."

In general, it would appear that ore to be handpicked should be washed and screened, and it seems doubtful whether belt picking will produce as good results as sorting on tables. By "as good" is not meant merely the quality of sorting, but the actual financial return. If the belt is conveying ore to mill bins and waste is being picked out, any inattention on the part of the pickers means waste rock in the mill feed, with consequent increased cost of milling and higher tailing losses. At any rate, a sorting belt should be slow moving, and a belt is in turn preferable to a revolving table, the difference in speed between the outer and inner lines of ore being unfavorable to good sorting. About 1 in. diameter appears to mark the lower limit of good hand sorting, and 3 to 9 in., depending on the ore, the upper limit.

The use of artificial light of various kinds may be an aid to hand sorting. Willemite may be readily picked out if illuminated by ultraviolet light. Metallic molybdenum may be readily distinguished from tantalum and stainless steel from nickel under illumination from the Westinghouse fluorescent lamps. Picking is best performed on rock freshly washed, and preferably screened.

Preliminary Washers.—Prior to final treatment, a certain rough separation by washing is often desirable. Preliminary washing machines include trough washers, log washers, washing pans, hydraulic giants, and trommels. They disintegrate the material and wash clay or fine stuff from the coarser portions.

The trough washer is simply a trough set on an incline, the material in the simplest form of washer being worked with hand tools. A heavy flow of water is admitted at the head end, usually from a feed box arranged to give a flow of water across the trough. The material to be washed is shoveled in at the tail end and worked up until coarse lumps and sand are removed at the head. Fine sand can be recovered from an external settling box and a clayey waste water led off. The machine is essentially a classifier.

It may be mounted on rockers and oscillated mechanically; or it may have power-driven stirrers, in which form it merges gradually into the log washer.

The log washer consists, essentially, of an iron or wooden trough with an inclination of 2.75 to 6 deg. in which revolves a thick wood or metal shaft carrying blades set obliquely to the axis so as to make a screw conveyor. The original machine was made with a log set with blades, hence the present name. As described by Richards,² the blades slowly convey the ore uphill against the washing current, discharging the lumps

¹"Ore Dressing," p. 35, 1923.

²"Ore Dressing" p. 184, 1909.

FREE-SETTLING VELOCITIES OF QUARTZ AND GALENA

Average Size of Grain in mm.	Average Velocity, mm. per Second.	Computed Velocity, mm. per Second. V.	Ratio Average Divided by Computed f.	Average Size of Grain in mm.	Average Velocity, mm. per Second.	Computed Velocity, mm. per Second. V.	Ratio Average Divided by Computed f.
Quartz Velocities.				Galena Velocities.			
11.93	393	621.5	0.6325	11.93	934	1235	0.7576
10.26	361	576	0.6269	10.26	865	1142	0.7576
8.65	340	532	0.6390	8.65	810	1052	0.7704
7.32	303	487.5	0.6219	7.32	729	968	0.7519
6.44	289	455.2	0.6349	6.44	680	906	0.7512
5.49	260	421	0.6173	5.49	631	836	0.7547
4.58	240	385	0.6242	4.58	558	765	0.7299
4.17	225	368	0.6116	4.17	538	731	0.7253
3.43	209	338.2	0.3179	3.43	513	671	0.7634
2.83	191	303	0.6305	2.83	450	607	0.7407
2.44	168	250.5	0.5983	2.44	420	557	0.7535
2.28	166.7	272	0.6135	2.28	42	539.5	0.8197
1.85	146.6	244.5	0.5988	1.85	370	486	0.7622
1.55	126.6	224.8	0.5634	1.55	330.5	445	0.7435
1.37	118.4	218.2	0.5435	1.37	295.1	418	0.7067
1.19	105.6	210.5	0.5082	1.19	270.1	360	0.6935
1.04	94.5	203.5	0.4604	1.04	252.5	364	0.6944
0.91	84.1	171.7	0.4902	0.91	227.5	341	0.6667
0.76	76.7	137	0.4890	0.76	207.8	311.5	0.6667
0.63	67.2	146	0.4608	0.63	192.8	283.5	0.6788
0.61	52.7	129	0.4090	0.61	160.4	255	0.6329
0.41	41.2	115.2	0.3571	0.41	126.1	229	0.5485
0.32	31.9	101.7	0.3135	0.32	103.1	202	0.5102
0.369	41.67	109.5	0.3802	0.345	125	209.5	0.5970
0.305	34.48	99.5	0.4202	0.279	111.1	189	0.5882
0.234	28.57	87.1	0.3279	0.215	88.5	165.8	0.5344
0.190	24.39	80.4	0.3030	0.160	74.6	145.5	0.5128
0.182	20.41	77.0	0.2657	0.127	62.9	127.2	0.4717
0.156	17.21	71.1	0.2427	0.106	52.6	116.2	0.4717
0.135	14.49	66.1	0.2193	0.0967	43.5	111	0.3922
0.126	12.05	64.0	0.1883	0.0839	37.0	103.5	0.3571
0.121	10.20	62.7	0.1626	0.0798	31.3	100.1	0.3125
0.116	8.55	61.3	0.1395	0.0714	26.3	95.2	0.2762
0.112	7.14	60.3	0.1183	0.0667	22.2	92.3	0.2404
0.0912	6.02	54.4	0.1111	0.0599	18.5	87.9	0.2105
0.0846	5.05	52.4	0.0904	0.0572	15.6	85.6	0.1821
0.0800	4.26	50.9	0.0840	0.0535	13.0	82.6	0.1575
0.0747	3.57	49.3	0.0725	0.0484	11.0	78.4	0.1403
0.0689	3.00	47.3	0.0635	0.0437	9.29	74.6	0.1241
0.0629	2.52	45.1	0.0558	0.0419	7.75	73.1	0.1059
0.0555	2.12	42.4	0.0500	0.0401	6.49	71.6	0.0906
0.0503	1.78	40.4	0.0441	0.0390	5.46	70.6	0.0774
0.0478	1.50	39.4	0.0380	0.0364	4.59	67.2	0.0653
0.0425	1.26	37.1	0.0340	0.0311	3.85	62.9	0.0614
0.0377	1.06	35	0.0303	0.0294	3.25	60.2	0.0541
0.0344	0.887	33.4	0.0265	0.0259	2.72	57.6	0.0473
0.0319	0.746	32.1	0.0230	0.0235	2.29	55.4	0.0413
0.0282	0.627	30.25	0.0207	0.0219	1.92	53.0	0.0362
0.0267	0.528	29.45	0.0179	0.0205	1.61	51.7	0.0312
0.0253	0.442	28.6	0.0155	0.0193	1.38	49.6	0.0277
0.0232	0.372	27.4	0.0136	0.0176	1.14	47.4	0.0241
0.0209	0.313	26.65	0.0120	0.0160	0.959	45.2	0.0212
0.0188	0.262	24.7	0.0106	0.0149	0.806	43.6	0.0185
0.0182	0.220	24.3	0.00905	0.0141	0.676	42.4	0.0159
0.0161	0.185	22.85	0.00819	0.0124	0.568	41.4	0.0137
0.0144	0.156	21.6	0.00723	0.0123	0.478	40.5	0.0118
0.0126	0.131	20.1	0.00709	0.0112	0.401	37.8	0.0106
0.00884	0.110	16.91	0.00649	0.00956	0.337	34.95	0.00965
0.00589	0.0924	13.81	0.00603	0.00760	0.283	31.8	0.00890
.....	0.00700	0.228	29.9	0.00797
.....	0.00593	0.200	27.5	0.00727
.....	0.00493	0.168	25.1	0.00670
.....	0.00431	0.141	23.42	0.00597
.....	0.00330	0.118	20.52	0.00575
.....	0.00152	0.0995	13.91	0.00715

at the upper end, the clay being disintegrated and floated to overflow at the lower end. The bottom may be an artificial semicylindrical bottom or a natural bottom formed by lumps of ore. There is a dam at the lower end; the upper end is open to allow free discharge of ore.

The ore is fed near the lower end and on the *rising* side of the log. The logs are usually driven from the upper end, in order to avoid the use of stuffing boxes. They vary in length from 10 to 30 ft., in width from 16 to 36 in. for single machines and 36 to 88 in. for double, and in depth from 8 to 50 in. They usually run in pairs at 12 to 18 r.p.m., a pair having 100 to 350 tons of capacity in product per 24 hr., or 2200 to 4400 tons of feed per 24 hr., at a usual cost of 10 to 50 cents per ton treated.

In the Thomas and the Turbo log washers the logs are geared together, so that they revolve in opposite directions, the rotation being such that the adjacent log faces are rising, the ore being fed in between them, on to the rising faces. It may be roughly calculated that a double washer of this type will take about 3 hp. and 150 gal. of water per minute, when handling 100 tons of ore per hour.

Iron logs are preferable to wooden ones as to operating costs, capacity, and repairs; in short, as to everything except first cost.

Wash trommels are hollow revolving cylinders or cones, usually with horizontal axes, which disintegrate and float the clayey matter while ore and water are passing through them. In the cylindrical form the forward passage of the ore is forced by oblique blades or by continuous screw threads. Friction wheels are rather better than spiders and central shafts for support and drive.

Washing pans are large circular iron pans, in which the ore is disintegrated by revolving blades, scrapers, or rollers.

The hydraulic giant is the well-known nozzle used to disintegrate gravel banks or the like, being particularly used in gold metallurgy.

Jigging.—Following hand sorting, jigging is the next cheapest, simplest, and crudest process. The foundation of the jigging process lies in two classes of phenomena, free and hindered settling. The first covers the fall of particles either in still water or against a rising upward current, where the motion is not interfered with by other particles. The second covers the movement of particles of mixed sizes, shapes, and densities in a crowded mass such that the particles are free to move, yet hinder each other's motion. In this mass they are sorted under the influence of a rising column of water. The general conditions of free and hindered settling have already been discussed on page 104.

The mode of operation of a fixed sieve jig with pure mineral fragments and screened within the limits imposed by hindered-settling ratios may be gained from the following elementary description of the separation action. If two grains (Fig. 1) *G* and *M* of different volumes and different specific gravities, *M* being of greater specific gravity and either the same size or smaller within the limits imposed by hindered settling, are at equal distance above the screen of the jig as at points *A* and *A* and they are given an upward pulsion by a fluid such as water, then they ought to arrive at the end of the pulsion at some such position as *BB*, but since both fall to some extent and *M* falls faster than *G* at the end of the upstroke, *G* will be in some position such as *C*, and *M* as some position as *D*. On the downward return following the upward movement, both grains will be aided by their downward impelling force, but since this gives to *M* the greater velocity, it will take some such position as *E* and *G*, *F*. By a number of cycles as just described, *M* will ultimately reach the screen, while *G* will still be at some distance above it. If now while falling relatively to one another, the two grains are subjected to a sidewise current, such as is produced by the flowage of ore and water through a jig, then evidently *G* will be more affected laterally than *M*, and may be caused to flow entirely away and to waste; while *M*, though not reach-

ing the screen in a position vertically under the point where it first enters the jiggling compartment where the screen is, but ahead of it, can be drawn through the holes in the screen or be removed by other means, and a separation is thus effected by the first principle enunciated at the beginning of this section.

The knowledge of the resistance offered by a viscous fluid to the passage of a solid body through it is far from being perfect. At high velocities, since there is a continuous pushing aside of the fluid by a mass of cross section A and with a velocity V , it would be expected that the energy consumed would be all or a portion of the energy given out by a flowing stream of cross section A and velocity V . In addition to this, there would be a force required to overcome the viscosity of the fluid, and since the surface of the grain and the velocity with which it is moving are the origin of this viscosity, it would be expected that it would be proportional to both of these factors.

Now at high velocity the retardation produced by viscosity is so small a portion of the total impedance that it can be neglected, and only the pressure produced by the moving grain need be considered. This is found to vary with the shape of the grain; *e.g.*, a smooth shuttle-shaped grain, moving downward with its long axis parallel to the direction of motion, offers no resistance or practically no resistance to the fluid, for the anterior pressure created by the down movement is balanced by a minus posterior pressure, and there exists no tendency whatever to set the fluid

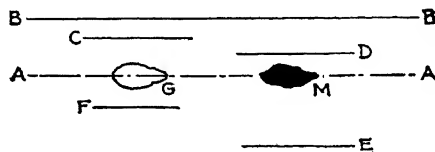


FIG. 1.—Theoretical settling in jiggling.

outside of such a body in motion. If, however, a flat grain is falling in a fluid, and with its long axis at right angles to the direction of fall, then since its advance causes the fluid to flow in a multitude of radial lines across the anterior face of the grain, and this sets the fluid in motion beyond the edge of the grain, the anterior pressure is largely used up in producing motion and very little is recovered in minus posterior pressure. (See table on page 128 for free settling velocities.)

The motion of the ore particles in jiggling may be brought about either by pulsion and suction of a moving water column, or by the motion of the ore bed through the water. While the latter is now chiefly a laboratory process, a hand jig is often a valuable adjunct to transient or isolated operations, and working drawings of a simple hand jig are given, as well as Richards's instructions for hand jiggling.

The products of a jig are as follows: (1) tailings, which form the top layer and are either skimmed off by hand or carried away by a water current; (2) concentrates; (3) hutch, the material going through the screen bed, this material then being either a fine concentrate or a middling.

(c) **Movable-sieve Hand Jigs.**—The method of working is to charge up the jig box with ore of proper size and depth. The coarser the ore, the deeper the whole bed may be, and the deeper the whole bed, the greater the output, but when too deep the separation by gravity is hindered. It is jigged with the proper amount of stroke and number of strokes per minute (the coarser the ore, the longer the stroke and the less the number per minute), giving a sharp downward motion to the screen to release the whole bed from it and so allow the ore particles to settle through the water under

hindered-settling conditions.

The motion should be stronger with coarse than fine ore. The return movement brings the water back through the screen and uses suction to draw down the fine concentrates. Experience only will give exact data on the speed, the amount of throw, and the number of strokes required for different ores.

When the jigging is finished, the lever is raised or lowered, as the case may be, and fastened to its hitching post *L*. The screen is thereby lifted out of water. The top layer is skimmed off with a short-handled hoe and thrown upon the tails heap. More ore is charged and the operation repeated until the concentrates have accumulated:

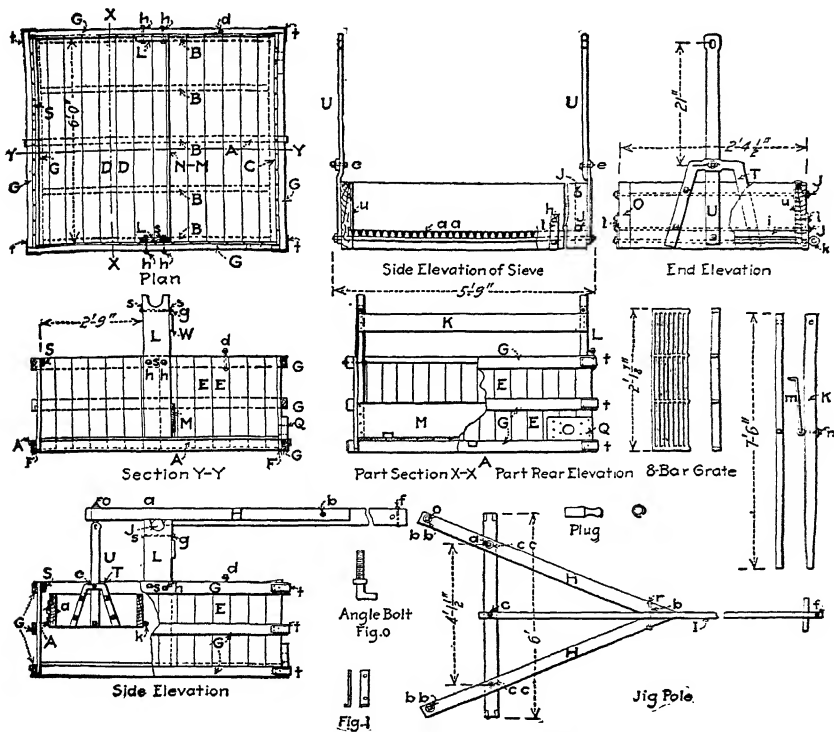


FIG. 2.—Details of simple hand jig.

then, after the top layer is removed, the middle portion is skimmed off, generally to be returned, and the bottom layer, which has accumulated up to 2 or 4 in. deep, is skimmed off as concentrates. The hutch product which accumulates vertically beneath the screen is shoveled out when sufficient material has accumulated, and the fine sludge which settles in the rear part of the jig tank is taken out separately. Some of the tanks are made large on purpose to secure this fine product, and in this case a partition, coming up two-thirds of the way, will keep the coarse hutch out of the fine sludge. The coarse concentrates and hutch products are generally treated again on a finishing jig with finer screen and make concentrates and hutch ready to ship. The sludge may be rich enough to ship, or it may need buddle treatment to

bring it up to the required standard. Where two minerals that belong to different markets, for example, galena and blende, are concentrated, they may be separated on a finishing jig.

Where recrushing of middlings is not to be resorted to, the jig, after several times having had tailings skimmed off from it and new ore charged, will be skimmed, yielding tailings or top layer, middlings to be returned, and coarse concentrates or bottom layer. The object of taking these middlings is in order that the concentrates may be freer from quartz and the tailings freer from ore. It also furnishes a layer on the sieve that prevents gangue from rattling down into the hutch while the next charge is being put on. After these middlings have been returned a few times, making an accumulation of them, the attendant will insensibly take off his tailings a little richer and his coarse concentrates a little poorer. This is his only way of disposing of the included grains for which his plant has no special provision.

A hydrant with water almost shut off, an overflow pipe, and a little settling tank may be provided for keeping the water at a constant level in the jig tank, or water may be added by a bucket from time to time. One or more holes are placed in the side of the tank, near the bottom, one below the other, for drawing off the water when it is desired to remove the sludge.

The labor required in all the mills is one man to a jig, which is high compared with machine jigs. The capacity given by Rittinger for his hand jig is 3 to 4 cu. ft. of ore per hour for each square foot of sieve surface. Hand jigs require little repairs. They can be put together anywhere with a saw, ax, chisel, auger, and a few simple iron pieces.

In practice, the walls of the jig box may or may not project above the water during all parts of the stroke. When it projects above, then suction is equal to pulsion, that is to say, just as much water will go down through the jig bed per stroke as rises up through it. When, however, the box is immersed, according to the amount of immersion, suction will be more or less diminished, leaving pulsion as much as before and giving a much softer and more open whole bed and one that would complete the separation into layers in a much shorter time. This is true because of the lift pump action of a jig which allows the water to rise more easily than to go down through a jig bed; here the water so pumped up flows over the sides when the jig is immersed. The latter method would be preferable for closely sized material, the former, probably, for mixed sizes.

The hand jig is a valuable means of testing the best conditions for treating any ore by jigging as it can be varied so easily and the results obtained so directly.

Continuous Movable Sieve Power Jigs.—These jigs are like the hand jigs in having a jigging tank screen and frame but differ from the hand jigs in having a stream of water introduced into the hutch. As actually run, most of them have a strong suction since most of the water that passes up through the jigging bed also returns through it. The Hancock jig of Australian invention is probably the leading example of this type. In this jig the screen has both a forward and upward and backward and downward motion. The jig is notable for its high capacity per square foot of floor space and the small amount of water used. B. J. Roberts of the Deister Machine Co. believes that the Hancock jig still has a place in rougher operations and that it would often pay to experiment with it, instead of rushing immediately to newer things.

Fixed-sieve Jigs.—These are of three classes: the Harz jig, in which the water is forced through the screen by the action of a piston or plunger in an adjacent compartment connected with the space below the screen; the accelerated jig in which the plunger is given more rapid motion during pulsion than during suction (these jigs are often incorrectly referred to as Harz jigs); the pulsion jig where a pulsating water

current is used and there is no suction. The illustration shows a common type of this jig as used in the Joplin district. A number of separations can be made simultaneously on complex minerals which is its main advantage.

Of the second type, the accelerated jigs, the New Century was possibly the most important. This had, in addition to a different plunger motion, a rubber flap valve around the entire piston which materially reduced the suction below the pulsion. The writer does not know of any New Century jigs in use at the present time.

✓ The Richards pulsator jig is the leading example of the third type. This jig is very remarkable for the capacity per square foot of floor space, but the very lack of bulk of the machine seems to prejudice the ordinary mill operative against it, and it

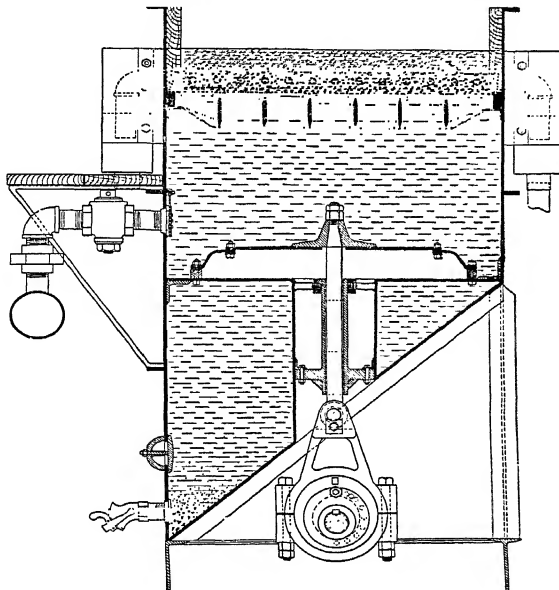


FIG. 3.—Bendelari diaphragm jig.

has never attained the success that apparently is indicated by its theoretical advantages.

The Bendelari diaphragm jig is the invention of F. N. Bendelari, president of the Consolidated Lead and Zinc Co., and is a recent improvement that has become very popular in the Tri-State lead and zinc field. The principal characteristic of the Bendelari jig is the use of diaphragms instead of plungers to cause pulsations. The line shaft and eccentrics are below the hutches. A tube through each hutch to the plate that carries the diaphragm permits connecting the eccentrics by means of a wrist pin and crosshead with the diaphragm head without using a stuffing box. The plate that carries the diaphragm is octagonal in shape which leaves the corners open and thus permits mineral to pass from the pulsation chamber into the still water in the hutch. Water is admitted below the sieves through check valves on the downstroke of the diaphragm. Stratification is rapid on these jigs, and different "draws" are used for different minerals. The mineral can be drawn from both sides of the bed.

Apart from the standard type where coarse concentrates are to be drawn from above the sieves, there is a type designed primarily for use on gold dredges and in closed circuits with a ball mill and classifier in gold mills which will treat down to 100 mesh.

The Pan-American pulsator jig, sometimes known as the Crangle jig, for gold and other free-metal concentration, operates on either a classified or an unclassified feed; it can be used on coarse classified pulps (+20 mesh) for concentration of heavy minerals, and is used in placer operations for cleaning rougher jig concentrates, for recovering gold and amalgam from cleanup sands, as a scavenger behind amalgamating equipment, and as a rougher in small-scale operations. The activation of the bed is by a high-frequency admission of water (400 to 600 times per minute) through a rubber

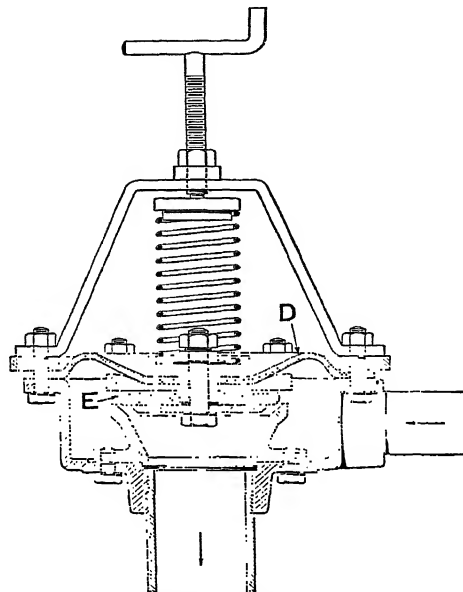


FIG. 4.—Pulsator, cross section showing water inlet at right and outlet into hutch below: *d*, flexible rubber diaphragm; *e*, valve.

valve under 5 to 10 lb. pressure. Dirty water can be used. It will make a good separation of gold or platinum from a black-sand concentrate.

The Conset (controlled settling) jig is another of the pulsation type. It was devised at the University of Minnesota, and the pulsations are due to inflation of a rubber tube in the hutch compartment by the admission of compressed air and its deflation by the weight of the water in the hutch when the air is shut off and the tube opened to the atmosphere. It has been experimented with considerably on iron ore, though there seems to be no reason why it should not also be used in concentrating nonferrous ores.

In regard to the general construction of jigs, there is probably nothing better than an ordinary pipe and plug out of which to make the spigots. For fine jigs, probably $\frac{1}{4}$ -in. pipe, for coarse jigs, a $\frac{1}{2}$ to 2-in. pipe with a wooden plug should be used. In making the ordinary Harz jig, the plunger should be the same size as the jig screen.

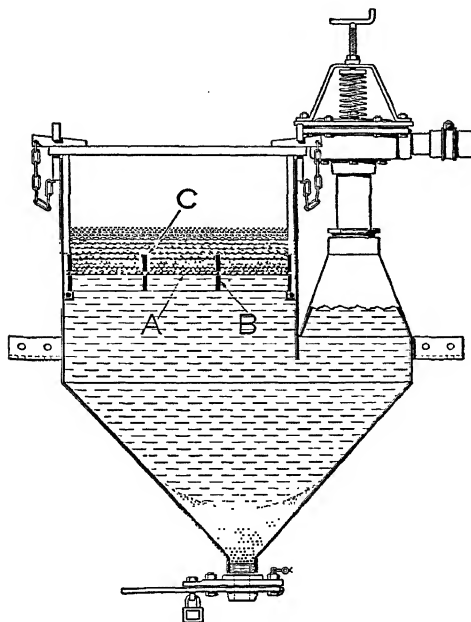


FIG. 5.—Pulsator jig, cross section normal to direction of pulp flow: *a*, screen, supporting shot bed; *b*, grids supporting screen; *c*, shot-retaining grid, which clamps down screen.

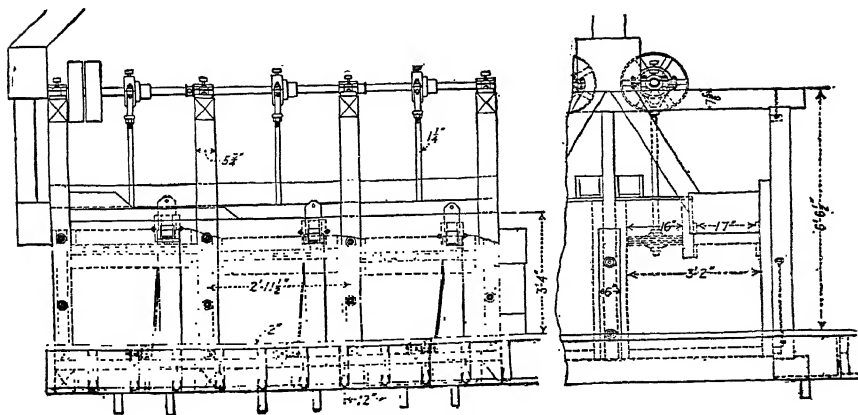


FIG. 6.—Harz jig.

The feed to jig should be steady. This is usually easily attained; as the feed comes from a revolving screen, a classifier, or another jig, it should be distributed uniformly across the feed end of the screen area.

The larger the sieve holes in proportion to the size of the feed, the more freely will fine grains reach the hutch and the less will the bed be clogged by their presence. A. Linkenbach recommends a proportion of 1:2 for grains above 5 mm., 1:1½ for 3-mm., and 3:4 for 2-mm. grains. In starting a jig, the screen bed should be filled with material of the same specific gravity as the concentrates, if such is obtainable. In hand-jigging operations it is customary, after discarding the tailings and cleaning out the concentrates, to use the middlings as the bed for the next charge. The amount of hutch product is regulated by the thickness and the mesh of the screen bed. The thicker the bed, the less freely concentrate passes into the hutch and the cleaner the hutch. With a thin bed the opposite is true. If the bottom bed is too thick, particles that should pass into the hutch are unable to do so and may be lost in the tailings.

Some method should be arranged to vary the throw of the jig plunger. The coarser and heavier the grains, the greater must be this throw. An ore making a large portion of concentrate will require a larger throw than an ore making a small amount of concentrate. Richards points out the following conclusions as to the proper method of jigging:

1. For jigging closely sized products, to get the highest speed of separation, use as little suction as the water supply will permit.
2. For jigging classified products where the hindered-settling ratio is equal to or larger than the interstitial ratio, in other words, where the concentrates are smaller than the interstices between the grains of the gangue, use suction.
3. For jigging classified products where the hindered-settling ratio is less than the interstitial ratio, in other words, where the concentrates are larger than the interstices between the grains of gangue, use suction.
4. For jigging mixed sizes and gravities, natural products, or products not closely sized, suction is suitable, as without suction the finer sizes are not saved.

The general practice of the day seems to tend toward a more general application of the English system, *i.e.*, toward the use of the jig in the treatment of unsized material instead of the hydraulic classifier. While the treatment of material sized between wide limits is possible and thoroughly practicable, still the advantages resulting from a preliminary sizing cannot be denied. In the English system itself, when the hutch products of one jig are treated upon another, we are making use of a preliminary sizing. Again, in order to jig an unsized product, suction is necessary to effect a separation, and suction, as has been stated previously, results in cutting down the capacity enormously.

The jig is receiving a new lease on life, as has been indicated above, in its use on gold dredges and in coarse grinding circuits to remove metallic gold and sulphides, and it has been introduced in the tailings launder of Arntfield Gold Mines of Quebec as a general scavenger. Jigs have an advantage over riffles in that they do not pack with black sand.

Heavy-medium Separation Processes.—The commercial heavy-medium separation processes depend on the fact that a suspension of a finely divided solid in water acts in many ways as does a liquid. The first application of this fact to ore dressing was by Chance, who about 1917 used a suspension of sand in water to separate coal from slate. Of course the fact that sand itself has a specific gravity of about only 2.6 made the use of this process impossible for metallurgical separations.

About 1934, experiments were begun in the use of galena in water as a suspension medium. A mixture of an effective specific gravity of about 3 is easily obtained, and

in 1936 the Mascot Mine of the American Zinc Co. placed the first heavy-medium separation plant in commercial service for the treatment of metallic ores. Later still, experiments were conducted with ferrosilicon with which it is possible to get a suspension with an effective specific gravity of over 3.2.

In general it is claimed that almost any ore that can be concentrated by jigging can be concentrated by heavy-medium separation processes. It also appears true that some ores can be treated by heavy-medium float-and-sink methods which cannot be treated by jigging.

At one plant, 2-in. ore is being treated economically while ore as fine as 100 mesh can also be treated. There appears to be a wide field of usefulness for heavy-medium separation processes as a preconcentration method for removing a large amount of the gangue at low cost after coarse crushing.

It is usually necessary to pass the feed through a scrubbing apparatus to remove any slimes, since they dilute the suspension media and increase their viscosity.

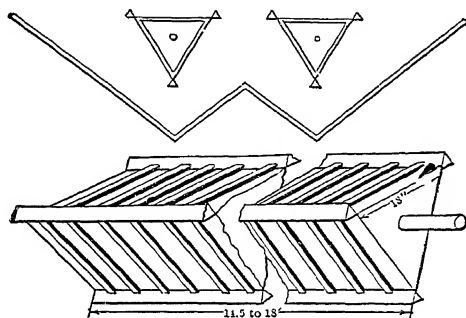


FIG. 7.—Revolving riffle strakes.

In the operation of the American Zinc Co.'s plant already referred to, the loss of galena was approximately 0.14 lb. per ton of material treated.¹ The process is simple in operation, the capital investment is low as are also the operating costs. A laboratory has been established by the American Cyanamid Co. devoted solely to this process.

One of the greatest items of expense in heavy-medium flotation-and-sink separation is the reagent. The comparative ease of recovery of galena and of ferrosilicon is the greatest reason for their use. Galena is recovered by tabling, and ferrosilicon is readily recovered by magnetic separation. The recovery of both of them is greatly facilitated by the removal of the fines from the material to be treated, and this alone would be a sufficient reason for this step in the process, although there are other advantages also.

Some rather unusual separations are made by float-and-sink procedure, for instance, Barton Mines, near North Creek, N.Y., is floating off a hornblende and feldspar tailing having a specific gravity of about 3 from a garnet concentrate having a specific gravity of 3.8 to 4.1.

Fine-sand Concentration.—The classic subdivision of fine-sand concentrators is into machines with the separating surface stationary, and those with the separating surface in motion. The first type is simple and inexpensive, and is based on the application of riffles to sluices, undercurrents, strakes, and carpet and blanket tables.

¹ "Some Recent Applications of Heavy Media Separation Processes" (Swenson, Faulkner, and Walker. *A.I.M.M.E. Tech. Publ.* 1600) published in *Mining Technology*, July, 1943.

Riffles as applied in sluices will be found discussed in Chap. XI of "Recovery of the Metals." When a grizzly is placed in a main sluice and the fines thus drawn off, the riffles for the fines are known as "undercurrents."

A number of riffles put together to form a polyhedron is known as a "strake." This apparatus can be revolved when the riffles are full, the riffle concentrate being thus emptied into a tank and a new set of riffles immediately being brought into place.

A modern substitute for the revolving strake is the so-called "blanket frame" of which so much has been heard in connection with the tin operations of Consolidated Mining and Smelting of Canada. The decks of these frames are made of $\frac{1}{4}$ -in. aluminum and are 6×6 ft. covered with rubber-covered cloth with a grid surface. The metal to be treated flows for 4 min. over the frame which is then tilted to 45 deg. and washed for 1 min.

In some mills, a table covered with carpet or blanket is placed after the amalgamating table to catch any rich material escaping the plates. This table covering is removed from time to time and carefully washed to remove any concentrate that has been caught.

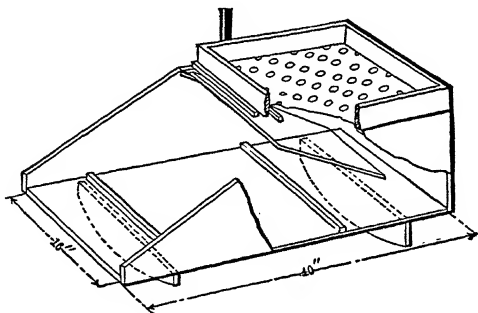


FIG. 8.—Cradle or rocker (with part of side cut away).

Of those machines treating fine sands where the separating surface itself moves, the simplest is the cradle or rocker.

The back, under the screen box shown in Fig. 8, is closed; the front is open. The bottom slopes from back to front. The illustration shows the sloping baffle board under the screen box to deliver all the material passing the screen to the very back of the rocker. There may be a blanket or carpet cover on this baffle. The bottom of the cradle is riffled. The apparatus is homemade and is still standard in placer work on a small scale.

The mechanical batea is a power-driven gold-washing pan, used for cleaning up the material from blankets, etc.

Sand Tables.—These devices have the separating surface in motion and have a continuous feed and discharge. Agitation is used to separate the feed into layers, the heavy concentrate on the table top then being advanced by the motion of the table in a direction either parallel or at right angles to the flow of the wash water, according to the type of table. In the Wilfley-type table, which is by all odds the most popular, the travel of the concentrate is theoretically at right angles to the flow of the wash water. In general, tables do their best work on material between 0.15 and 2.0 mm. in size.

Of the end-shake tables having motion of concentrates parallel to the flow of the wash water, the Gilpin County bumping table, so called because of its former great

popularity in that district of Colorado, is the best example. Richards describes it thus:

"The table is smooth, of the end-bump type, and a spring, cam, and bumping post furnish the bump. This agitation separates the material into layers. The bumping action propels the ore particles up the slope, but wash water coming down the slope carries away the gangue and leaves the heavy mineral behind. The table receives 120 to 180 bumps per minute. The magnitude of the movement is $1\frac{1}{2}$ to 3 in., depending upon the character of the material. These tables must be fed with unclassified pulp, and a single table will treat 4 to 5 tons per 24 hr."

The tables of the second class, the motion of the concentrates being at right angles to the travel of the wash water and tailings, are sometimes known as "jerkling tables"; they make use of the same principles as do the bumping tables, but in addition have riffle cleats or grooves to facilitate the concentration. These riffles may be tapered, varied in many ways: the ends in a straight line or otherwise, etc. The shaking

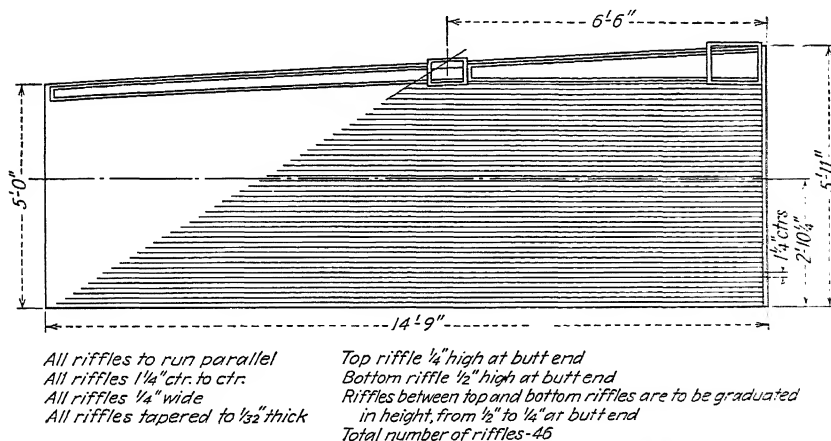


FIG. 9.—Wilfley standard No. 6 table. (Courtesy of Mine & Smelter Supply Co.)

motion may be by eccentrics; by eccentrics, toggles, and springs; by spring and bumping post; etc., and is parallel to the riffle grooves. In all cases the backward motion is the more rapid in order to drive the concentrate grains lying next to the table top along the riffles under the waste grains above, which are washed down the table top across the riffles. The line along which the concentrate and tailing grains part company is a diagonal or slightly curved line, and theoretically the line of the ends of the riffles will be identical with it, or else a flexible bend in the table deck itself marks where this line should be.

A motion obtained by cams, toggles, and springs in this type of table is preferable to the bumping-post motion because of the effect on the mill building, so that the Wilfley type and its variations are the most popular today.

Wilfley and Wilfley-type Tables.—The original Wilfley table was patented in 1895, and the various tables in use today embody the Wilfley features or those taken by him from Rittinger. The deck is usually covered with linoleum, and the riffles are usually of wood.

The slope of the deck from the wash-water box and feed-box side to the tailings side opposite can be varied so that the line of separation of concentrates and tailings

can be held coincident with the riffle-tip ends. The cleats may be varied as to their dimensions, both as between tables for various ores and on the same table. If varied on the same table, the lower ones will be the larger. About $\frac{1}{2}$ -in. square is the largest size that seems good.

While the illustration shows no riffles over a portion of the table, which then becomes effective in cleaning the concentrates, with a heavily mineralized feed the riffles may extend all the way to the concentrate side, in which case the concentrates will probably be cleaned on another apparatus.

Some tables have a flexible joint in them at the ends of the riffles so that the cleaning portion of the table can be tilted into a different plane from the riffled portion. In some tables the riffles are V-shaped or wedge-shaped, and in some the riffles are divided into groups, a high cleat separating each group.

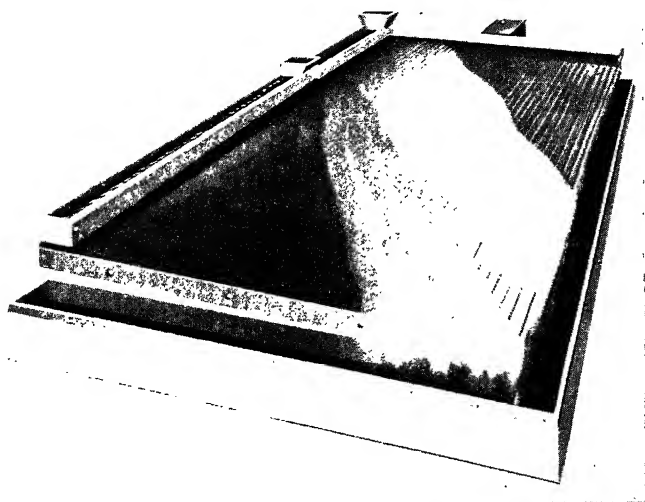


FIG. 10.—Concentration of material on Wilfley table deck.

In the Wilfley deck a rim extends along the feed and drive sides, but the concentrate and tailings sides are open. The tailings lip or launder along the tailings side ordinarily does not extend quite to the corner between the tailings and concentrate side, a special launder on the last 8 to 12 in. collecting a middling. In the same way, the product on the last few inches of the concentrate side may sometimes be thrown into middlings, but table adjustment should obviate this.

Customary practice is to lead these middlings back over the same table, but the author believes superior results are obtained in mills having many tables, if all the middlings are diverted to a single table, and in such case it should be made the subject of experiment as to whether there should not be a slight recrushing before retreatment.

Feed.—The feed is through a perforated box, which, on a 16-ft. table, will occupy about 4 ft. of the feed side, the remainder being occupied by the wash-water box, the best arrangement being one continuous box with a movable partition enabling any division desired between the two portions.

tinguished by curved riffles throwing the concentrates away from the tailings edge. The stroke can be varied from $\frac{7}{16}$ to $1\frac{3}{16}$ in., and the deck moves in a curvilinear path, which is supposed to throw the concentrate away from the upper side of the riffle cleats. The smaller tables are also still manufactured.

A table will treat 5 tons of very fine material to 40 tons of coarse per day of 24 hr., using $\frac{1}{4}$ to 1 hp., and taking 12 to 35 gal. of water per minute. The slope will be anywhere from 0 to 3 deg. up from the drive-end to the concentrate discharge. This must be determined by experiment. While materials as coarse as $\frac{1}{4}$ in. and as fine as 200 mesh have been reported as successfully treated on a table, most feeds run from 16 to 120 mesh. From a consideration of the theory of hindered-settling ratios, the author believes that a sized feed should be used, though many will be found to disagree and argue either for a "natural" feed (unscreened and unclassified) or for a feed classified by a hydraulic classifier. Richards believes in the last. The improvements in table construction and operation have practically killed the use of vanners, described below.

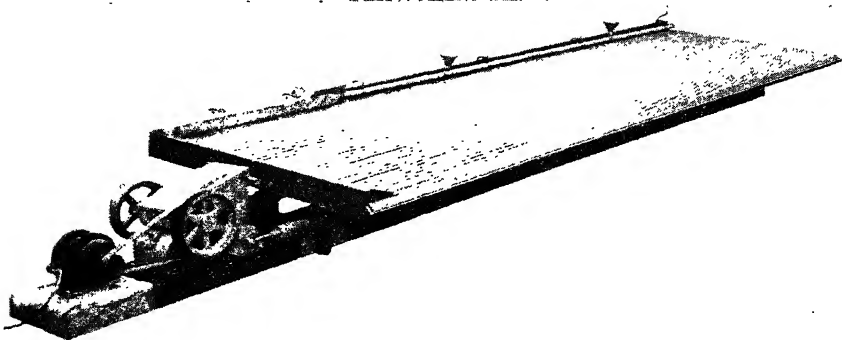


FIG. 12.—Overstrom giant table.

Vanners.—The vanner depends for its action on the slow travel up a slope of an endless belt, the concentrates being carried up and over the top and there washed off, while the tailings are washed down the belt and off at the bottom. The mechanical motion may be side shake, end shake, or undulatory. Vanners are essentially slime-treating machines.

The belts may be rubber or canvas, and if of rubber, may either be plain or lightly ridged, and with various types of edge or lip. The vanner cannot make more than one separation and is complex and needs skilled supervision.

The leading types of side-shake vanners were the Frue and the Isbell; the Triumph was the leading end-shake machine; while the Johnson was arranged to give an undulatory motion to the belt. There are a few vanners known to be running, but in general the notes concerning this form of equipment are of more historical than operating interest.

Slime Tables.—The vanner was largely forced out by the slime table and by the rise of the flotation process. The Deister was probably the leading slime table. It was a riffled table employing the same motion as the sand table, but much of the separation and stratifying was done in a pool of slimy water in the feed and drive corner of the table.

The greater use of well-classified feed and the improvements in sand-table design have in turn rendered the slime table nearly obsolete. There are, however, some known still to be running, and their performance leads one to believe that possibly there are many places in which these tables might be profitably used.

Slime Treatment.—Apart from the use of the ordinary concentrating tables, and the few surviving vanners, canvas tables of various sorts are also much used. Number 6 duck laid with the warp at right angles to the flow of water is laid at an angle of $1\frac{1}{4}$ to $1\frac{1}{2}$ in. per ft. and fed with a pulp containing 10 to 16 per cent by weight of solids and about 2 lb. of water per square foot of treating surface used to wash the quartz away.

A round table of concrete is also used, the roughness of the concrete holding the concentrate, and Utah copper is still using about 30,000 sq. ft. of burlap in its tailings launders to catch the last heavy material in its tailings. The burlap is burned monthly and the ashes sent to the smelting furnaces.

But after all, flotation appears to be the best method for cleaning a sulphide slime.

Magnetic Separation.—All substances are either attracted or repelled by a magnet. For convenience, the permeability of air is taken as unity and all other substances are compared with it. The great majority of substances are diamagnetic, *i.e.*, repelled by a magnet, though the permeability of those most repelled is much less than that of the strong paramagnetics (substances attracted by a magnet). The most strongly repelled substance known, bismuth, has a permeability of -0.9998 , while iron may run over $+2000$. The principal paramagnetics are iron, nickel, cobalt, manganese, chromium, cerium, titanium, palladium, platinum, osmium, the oxides of iron, oxygen, and certain alloys of manganese, aluminum, and copper. The principal diamagnetics are bismuth, antimony, zinc, tin, cadmium, mercury, lead, silver, copper, gold, sulphur, arsenic, uranium, rhodium, iridium, and tungsten.

Minerals that are attracted, with their relative permeabilities are: magnetite, 800; franklinite, 710; ilmenite, 495; pyrrhotite, 140; biotite, 64; siderite, 36; zircon, 20; limonite, 17; pyrolusite, 14; and roughly in order of decreasing permeability, mangonite, calamine, garnet, quartz, rutile, cerussite, cerargyrite, argentite, orpiment, pyrite, sphalerite, molybdenite, dolomite, bornite, apatite, willemitte, tetrahedrite, talc, arsenopyrite, magnesite, chalcosite, cuprite, smithsonite, orthoclase, stibnite, cryolite, enargite, senarmontite, galena, niccolite, calcite, and witherite, the last having a permeability of 0.4 as compared with 2000 for ordinary iron.

The foregoing list is furnished by the Dings Magnetic Separator Co., but attention is called to the fact that the list is only an approximation and that the same substance from different geographical localities varies widely in its magnetic properties. Frantz and Jarman in a paper before the American Institute of Mining Engineers in 1932 suggested that this might be due to different quantities of isotopes in minerals from different localities.

Magnetic separation has made great advances in recent years due in part to the knowledge of how to use fields of magnetic force of much greater intensity than formerly and in part to better mechanical devices. In general, the principle of magnetic separation is to feed the ore only a few grains deep into a magnetic field to attract the paramagnetic materials away from the gangue and then carry the particles thus attracted outside the magnetic field. Commercially, there are four important methods by which this is done. The material to be treated may be spread in a thin layer on a horizontal belt, and the highly magnetic materials may be attracted away from the less magnetic material against the force of gravity and carried away from the magnetic field by a belt or belts running at right angles to the line of travel of the feed belt. This is the principle of the Rowand-Wetherill separator. Or the belt may run in the same direction as the feed belt but parallel to it, under a series of magnets,

thus carrying the magnetic material along until it is outside the field of the lower magnets and then dropped; or the feed belt may simply go around a magnetic pulley, the nonmagnetic material dropping vertically from the belt at the extremity of its travel while the magnetic material is carried around clinging to the belt until the belt runs off the pulley. This is the principle of the widely known Dings magnetic pulley which is used for the removal of tramp iron. A variation of this idea is to have a drum of practically nonmagnetic material such as brass running around magnets that occupy only a portion of its circumference. The less magnetic material drops vertically from the drum at the extremity of its travel, while the magnetic material is carried around until there are no magnets under it when it drops into the concentration chute. There is finally a type that depends on inducing a high magnetic concentration in a roller which deflects the magnetic material falling in a down stream near it. By the use of an adjustable lip, a cut can be made in this stream between the magnetic and the nonmagnetic material, and it is obvious that a particle of less magnetic permeability can be deflected from a falling stream than can be picked up by a magnet against the operation of the force of gravity.

In this induction separator, a cylindrical armature made up of thin laminated disks of a special annealed wrought iron and mounted on a steel shaft revolves horizontally between the pole pieces of a large inverted horseshoe field magnet. By induction, the magnetism of the pole pieces causes magnetic poles to appear on either side of the surface of the armature. Material to be separated is fed from the hopper on the top of the revolving armature and is carried by its movement around the armature. At the bottom position where the polarity changes, the last of the material clinging to the armature will fall off as there is no attraction there whatever. The magnetic particles will fall off through an arc of 90 deg. in the order of their attractability, and by suitably arranged hoppers below, separations can be made taking advantage of attractive differences. The magnetic material from one roller may be re-treated by one or more succeeding rollers, and a very effective separation is obtained.

The material to be treated in magnetic separators is preferably about 12 to 80 mesh, though under favorable conditions material as fine as 300 mesh is said to have been treated. The difficulty with fine materials is to furnish a feed that will be only a few grains deep.

If any material that is being tested to see whether it is amenable to magnetic concentration does not respond to the treatment in its raw state, it should be tested after a reducing and an oxidizing roast. An oxidizing roast is more likely to change a nonmagnetic to a magnetic body than is a reducing roast, hematite and limonite being exceptions.

Magnetic separation has solved many problems, but it must be used with some care. For instance, magnetic pulleys were installed in one of the mills at Kirkland Lake, Ont., in order to remove finely divided iron that was a cyanicide. They removed the iron, but it was found that there was enough gold with the iron more than to offset the value of the cyanide that was saved. Removal of magnetite from a black sand concentrate must be done with care as it is possible that platinum, palladium, and osmium will be removed with the magnetite.

The same machine may be used to make several separations by varying the current and the speed of travel of the feed and take-off mechanism. Thus, on a black sand concentrate the magnetite may first be taken out with very little else by using a comparatively weak current and running the feed at high speed, then other separations made at increasing field intensities and slower speeds. Ordinarily the feed to a magnetic separator must be bone dry.

Electrostatic Methods.—Electrostatic methods of concentration depend on the fact that two substances having unlike charges of electricity will attract each other,

and those having like charges, will repel. Some substances will allow a current of electricity to pass freely through them; these are known as "conductors." Others will not allow a current to pass freely; these are known as "nonconductors," or dielectrics.

The attraction or repulsion varies directly as the mass and inversely as the square of the distance.

Every mineral will conduct electricity or take an electric charge on its surface if brought into contact with a surface charged at a high enough potential, but there is a great difference in their relative conductiveness. Among the good conductors are native minerals, metallic sulphides, arsenides, antimonides, selenides and tellurides, garnet, magnetite, hematite, hornblende, biotite, and graphite; among the poor conductors may be listed quartz, quartzite, sandstone, granite, porphyry, feldspar, epidote, calamine, spinel slate, fluorspar, monazite, calcite, dolomite, barite, siderite, sphalerite, gypsum, and most silicates, carbonates, and sulphates.

If a mixture of minerals in the neutral state is allowed to fall on a highly charged surface, most of the constituents will be repelled more or less strongly from this surface and a separation can be made by using knife-edge splitting devices in conjunction with a charged rotor. However, some of the common minerals are also repelled from a negative electrode but not from a positive electrode. These are silica, bauxite, smithsonite, oligoclase, enstatite, pyroxene, amphibole-hornblende, zircon, tourmaline, kaolinite, and sphalerite.

Among those always repelled from a positive electrode, but not from a negative, are calcite, sulphur, cryolite, dolomite, magnesite, aragonite, chrysolite, topaz, muscovite, serpentine, apatite, anhydrite, bituminous coal, alumina oxide (abrasive), and zircon from beach sands.¹

All other common substances than the above mentioned are invariably repelled from both the positive and negative electrodes. It is to be noted that sulphur is the only element that does not fall in this classification.

The commonly used Huff machine depends on a charged rotor. Sutton, Steele, and Steele feed the material to be treated on a smooth-faced rotor about 6 in. in diameter after warming it to about 125 to 150°F. Suspended above the point of feed and parallel to the rotor is a rod containing a number of needle points. The rod is charged positively, using high-voltage low-amperage current. All the particles receive a charge from these points. Since the rotor is grounded, all the particles tend to cling to the rotor. The better conductors lose their charge soonest and drop quickest from the rotor, while the poorer conductors cling longer.²

Electrical machines in general and particularly insulators have been greatly improved within the last few years so that electrostatic machines are far more reliable than in former years. The material must be fairly fine and must be warm and dry, so there is always a dust problem. The fact that the feed can be only a few particles deep means an expensive installation for the capacity. Jarman gives the cost of an electrostatic separator at \$3000 to \$4000 per ton-hour.

So long as the material itself is dry and warm, the humidity of the atmosphere seems to make no difference nor does internal moisture, provided the surface is dry during the time that the particles are in the electrostatic feed. Close sizing of the feed is essential. About 10 mesh appears to be the limit of size for coarse material and about 150 mesh for fine, though separations are said to have been successful with material as fine as 300 mesh where it was granular in its character. Roasting will sometimes render an ore amenable to electrostatic treatment when it is not so originally. Thus in connection with the electrostatic cleaning of North Carolina kyanite concen-

¹ JOHNSON, A.I.M.M.E. *Tech. Paper* 877, February meeting, 1938.

² JARMAN, A.I.M.M.E. *Tech. Publ.* 959, October meeting, 1937.

trates, V. L. Mattson found it necessary to heat and cool the material before treating to eliminate undesirable complications which he attributed to radioactive elements.¹

The framework of electrostatic machines should be grounded as should also the neutral electrodes of the Huff machine. Electrodes must be kept free from grease and dirt.

Separation by Blowing.—Separation by air has been used for many years in the seed industry, and Sutton, Steele, and Steele have applied the principles of the air separator to metallurgy. In general, the principle of the air table is the same as that of the ordinary wet concentrating table, but it has the disadvantage that the free settling ratio of different minerals are less in air than in water, *e.g.*, the settling ratio

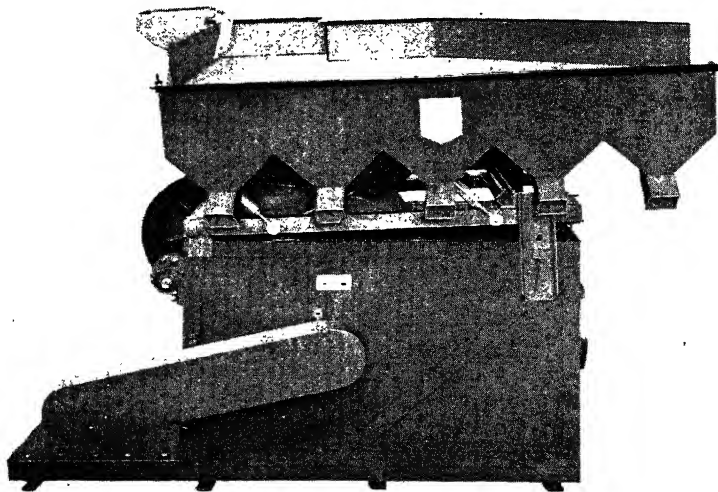


FIG. 13.—Sutton, Steele & Steele air table.

of galena in quantity is 4.01 in water and only 2.08 in air. This means that closer sizing of the feed is necessary for successful air-table operation than for wet table. The air table has the great advantage that the total weight of material on the table top is a working load instead of carrying a great interfering weight of water on the table top as is the case with the wet concentrating table.

Very good work is done by this table on such separations as asbestos fiber from its gangue; on the separation of wolframite; on the treatment of scheelite ores; such separations as rutile from zircon and on the separation of metal from oxide drosses. G. N. Jarman states that the air table has been used on separations as fine as minus 250 to plus 400 mesh, though 100 mesh is the minimum size usually treated, and he prefers +65 to finer material. In one exceptional case he tabled $\frac{5}{8}$ -in. material, so the air table has a wide possible range of size. Separations on carefully sized material are said to have been made on a difference of 10 per cent in specific gravity. Jarman gives the installation cost of an air separator as \$350 to \$1000 per ton-hour, and the operating cost as the cost of screening plus 10 to 30 cents per ton.

¹ *Mining Met.*, July, 1935, p. 295.

The machine consists of a flat deck mounted a short distance above the floor level by means of rocking legs. The deck surface is cloth, and for treating coarse materials more open cloth is used than for fine materials.

A reciprocating motion is given to the deck by an ordinary eccentric, and since the rocking legs incline toward the eccentric or head-motion end of the machine, at each forward stroke the deck rises slightly and falls back an equal amount on the return giving the necessary differential for advancing feed along the deck. An ordinary centrifugal fan is built into the frame of the machine, and a flexible connection leads to an air chest under the cloth-mounted deck. Regulation of the air through the cloth is obtained by a gate on the suction side of the fan. A series of shallow wooden riffle strips are secured to the deck parallel to the line of the pulsions. Material is fed at the upper right-hand corner looking down the deck from the head-motion end. It spreads out over the deck under the combined influence of the advancing motion and the slightly transverse inclination given the deck. By interstitial action due to the shaking of the deck, explained elsewhere in this work, the small particles work down through the bed of material on the deck and advance to the lower end where they are discharged. The large particles that stay on top of the bed move under the influence of the transverse slope of the deck more nearly across the table and discharge over the long side. The deck is 10 ft. long and 5 ft. wide.

The riffles assist in guiding the small particles to the end of the deck or, what is the same thing, prevent them from working transversely across the deck with the large particles. The riffles are of such height that they do not interfere very much with the transverse travel of the large particles. If the small particles are of greater specific gravity than the large, there will be separation according to gravity. If the fragments or particles are all of the same composition, the machine will effect a grading. The function of the air blown through the cloth is to keep the bed loose so that there will be no packing, thus permitting interstitial action to have full play. It has been stated that the air actually raises and holds in a state of balance the larger particles. It is true that with fine materials the air creates so much looseness in the bed that the upper layers appear to be in a state of suspension, and with closely sized material such a conception of the principle of operation may have some modicum of truth in it.

Separation Depending upon Sliding Friction.—While devices based on differential sliding friction are used in the anthracite industry to separate coal and slate, no application is known to the metallurgical field.

Testing in Concentration Operations.—For testing small samples and for checking the results in concentration operations, the Superpanner and the Infrasizer, both invented by Dr. H. E. T. Haultain of the University of Toronto, are worthy of attention.

The *Infrasizer* is an air elutriator and in its present form consists of six stainless steel cones, ranging in their largest diameters from $2\frac{1}{2}$ to 14 in., and in length from 4 ft. 6 in. to 2 ft. 6 in. These are connected in series by rubber tubing, the largest cone discharging into a dust-collecting bag. At the lower end of each cone is a rubber plug and ball, which is an essential feature of the apparatus.

There is also a panel board, which automatically controls the volume of air admitted to the system and by the aid of time switches shuts off the air at a pre-determined time.

The sample is placed in the smallest cone and is distributed through the series according to size. The average diameters of particles in succeeding cones varying as the square root of 2.

The great difficulty in all size analysis of fine powders, whether by wet or dry methods, lies in the fact that the finest particles adhere to the coarser particles and to

each other. The essential feature of the Infrasizer is the scrubbing action that takes place at the lower end of the cones. In each cone is a removable rubber plug, coneshaped inside, in which sits a ball. For this purpose, a steel ball $\frac{5}{8}$ in. in diameter, glass marbles, and ping-pong balls have been used, but for general purposes a golf ball is now found most suitable. The entering air, with its load of sample, lifts the ball and travels past it in a thin high-velocity stream with much eddying as it passes the ball. Particles or flocs not escaping through the top of the cone fall down the side of the cone and come under the influence of this eddying. The result is a remarkably perfect scrubbing, freeing the particles from each other.

The time required for a satisfactory size analysis depends on the type of material. Extremely fine mill tailings, especially where lime is present, may require twice as long as more easily scrubbed material. For ordinary mill tailings and such products as cement, the time is usually $\frac{1}{2}$ hr. for 50 g., 1 hr. for 100 g., etc. Samples up to 500 g. can be sized, requiring 5 hr. running time for this amount.

The *Superpanner*, in its present form, performs the same functions as the gold pan, the prospector's horn, the vanning shovel, the batea, the vanning plaque, or the sichertrog, but substitutes continuous controlled mechanical operation for hand operation. It consists of a trough, 30 in. long, and the operating mechanism with its adjustments. The pan (trough) is suitably mounted and connected to permit a longitudinal backward and forward movement, ending in a bump as in the old Gilpin County bumper, and a gentle to-and-fro side motion. The following adjustments can be readily made while in operation: the slope of the pan; the intensity of the end bump; the length of stroke accompanying the end bump; the number of strokes per minute; the amplitude of the side shake separately for each end of the pan; the amount of wash water; the depth of the pool in the rear end of the pan.

The accumulating wash water is removed by continuous suction and may be caught in a vacuum bottle so that none of the products need be lost. The concentrate can be removed from time to time by a suction pipette. It will operate satisfactorily with as little as 5 g. of material and will treat material as coarse as 14 mesh. It will recover clean pyrite from the Infrasizer $1\frac{1}{2}$ -micron product and will separate gold or heavy minerals, such as tellurides, from material finer than 10 microns. Of course, it does better work with sized products and makes useful separations of minerals with small differences in specific gravity.

It is particularly useful in gold milling to recover clean pyrite free from fine gold and tellurides and to recover clean gangue minerals, thus determining the distribution of the gold. It will separate pyrite from arsenopyrite. Usefully accurate quantitative determinations can be made of the amount of sulphides present. As a check on flotation cells, it quickly determines the cleanness of concentrates, middlings, or tailings. It is useful for the recovery for identification of the heavier minerals existing in very small quantities. It will readily recover tellurides when occurring in the proportion of 1 part in 10 million. Geologists have found this useful when studying the presence of minute quantities of heavier minerals in rock. Both the *Superpanner* and the *Infrasizer* are distributed by Infrasizers Limited of Toronto.

PART III. FLOTATION

By S. J. SWAINSON¹

General.—The flotation process is the most widely used method of wet concentration of ores for separating the valuable constituents from worthless gangues. The process is primarily based on surface phenomena, and the specific gravity of the mineral particles plays little or no part in the separation.

In the practical utilization of these phenomena, when air bubbles are introduced into a properly ground and treated ore pulp, particles of certain minerals will become attached to the bubbles, while other minerals will not so adhere and will remain in suspension in the pulp. The mineral-laden bubbles rise to the surface of the pulp as froth, or "concentrate," which overflows the flotation machine by gravity or may be removed by mechanically operated skimmers. The nonfloatable portion, or "tailing," discharges from the end of the machine through openings provided for the purpose.

In general, flotation operates by virtue of a number of interrelated physiochemical factors, including

1. The relative degree of wettability with water of different mineral particles.
2. The relative degree of readiness with which the particles of different minerals adhere to air bubbles in an ore pulp.
3. The affinity of certain mineral particles for certain types of chemical compounds, or "reagents."

Various theories have been advanced to explain the mechanism of flotation, and opinion is divided between adsorption and chemical reaction. There is much evidence to substantiate the belief that both adsorption and chemical reaction play their part in every flotation separation.

The physiochemical concept of the mechanism of flotation takes into consideration such properties of physical chemistry as surface energy, surface tension, adsorption, contact angle, polarity, surface condition, and reactivity, as well as surface chemistry.

In general, three different varieties of flotation processes are possible:

1. Film flotation.
2. Bulk-oil flotation.
3. Froth flotation.

In the early developments of the flotation processes, film flotation and bulk-oil flotation were commercially used to some extent, but today only froth flotation is of importance. In simplest terms, froth flotation necessitates the addition of specific chemical compounds, or reagents, to an ore pulp that will

1. Produce a froth of desired character.
2. Modify the surfaces of the mineral or minerals that are to be floated so that these minerals will adhere to the froth bubbles.
3. Modify the surfaces of the unwanted minerals so that they will not adhere to the bubbles.

Historical.—The history of the development of the flotation process makes interesting reading. The earliest record of the principles of flotation being applied to an ore dates back to 1860 when William Haynes in England proposed as a method of beneficiation the admixture of coal tar or resin with an ore followed by separation of

¹ Director, Mineral Dressing Laboratory, American Cyanamid Co., Stamford, Conn.

the valuable constituents from the gangue by frictional trituration with water. Later on, in 1886, Carrie Everson in the United States disclosed in U.S. patent 348157 the use of fats or oils to buoy sulphides and thus separate them from worthless gangue, the selectivity of the separation being assisted by the addition of acids and acid salts or neutral salts.

In the foregoing "bulk-oil" methods and later improvements of this form of flotation by Elmore, Cattermole, and Macquisten, the wet ground ore was mixed with a comparatively large proportion of a viscous oil, and the desired result was to induce the mineral particles to adhere to the oil globules; the thick mixture rising to the top of the pulp owing to its lighter specific gravity was removed as a concentrate. It is to be noted that air was not intentionally introduced to assist the separation of the oiled particles in these processes. Large quantities of oil were necessary for the separation. Recoveries were poor, and the grade of concentrate produced by these methods was low as measured by modern froth-flotation standards. Bulk-oil flotation was never a commercial success, although tried at several plants throughout the world.

Following this early era of bulk-oil flotation, it was discovered that improved results could be obtained by the introduction of gas bubbles into the ore pulp in order to assist the separation of sulphide particles from nonsulphide mineral particles. At the turn of the century, processes were developed by Delprat and Potter utilizing gaseous media generated by the reaction of acids on carbonates and sulphides in the ore. Later on, Froment introduced his process for introduction of a gas into a freely flowing oiled pulp in order to assist flotation and effect a reduction in the amount of oil normally required in bulk-oil flotation methods.

Following the improvements obtained by these early investigators, and fuller recognition of the value of gas or air as an aid in the separation of minerals by oil flotation, the next advance resulted from the developments of Elmore in 1904, Sulman, Picard, and Ballot in 1906, and by Hoover in 1910 wherein gas or air was introduced to the ore pulp by vacuum generation or under pressure or by means of mechanical agitation. Introduction of air to the pulp was accompanied by a very marked reduction in the amount of oils required to float the minerals, but consumptions were still high and selectivity was inefficient. However, in 1909, Greenway, Sulman, and Higgins advanced the art through their discovery of soluble frothing agents such as ketones and alcohols, which permitted still further reduction of the quantity of oils formerly required for flotation.

From 1913 to 1922, flotation may be said to have first attained commercial success, particularly in the United States, the first commercial plant of importance being that of the Butte & Superior Co. placed in operation in 1913. A wide variety of oils, such as wood and coal-tar cresotes, were used as collectors in conjunction with frothers, such as pine oils and rosin oils, for the sulphide minerals. It is to be noted that all these collectors were of indefinite chemical composition. Bulk sulphide concentrates were recovered inasmuch as it was not possible to separate one sulphide mineral from another with the known flotation reagents of that era. Sulphuric acid was commonly used to assist the work of the promoters.

This period also witnessed the development of many types of flotation machines, including the Minerals Separation, Janney, Callow, K & K, and Welch.

The present modern, or "chemical-flotation," period originated with the discoveries of Perkins, as disclosed in his U.S. patent 1364043, in 1921, of the effectiveness of definite chemical compounds of nonoleaginous nature as promoters for sulphide minerals. These compounds were organic trivalent nitrogen and divalent sulphur compounds of slight solubility, such as alphanaphthylamine and thiocarbanilide. They do not include the completely water-soluble xanthates patented by Keller and Lewis in 1924 and disclosed in U.S. patents 1554216 and 1554220.

Since the discoveries of Perkins and Keller, numerous important advances in the art have occurred, among which the following may be listed:

1922—Cyanide as a depressant for pyrite and sphalerite in alkaline solutions, discovered by Sheridan and Griswold (U.S. patent 1421585).

1923—Sodium sulphite as a depressant for sphalerite, discovered by Bragg (U.S. patent 1478697).

1924—Discovery by Sulman and Edser of the usefulness of fatty acids and soaps to float oxidized base-metal minerals.

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1935—Anionic-type wetting agents, discovered by Lenher (U.S. patent 2012609).

1938—First use of cationic-type wetting agents to float siliceous mineral, Lenher (U.S. patent 2132902).

Theory.—These advances resulted in large measure from a better understanding of the underlying principles of froth flotation. In this regard, Langmuir was one of the first, if not *the* first, to explain the chemistry of the process in his paper "The Mechanism of the Surface Phenomenon of Flotation."¹ The essentials of the process are summed up by him as follows:

"The surface phenomena of flotation may be divided roughly into three classes:

1. Formation and properties of the froth.
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Dr. L. J. Christmann, in a classic dissertation on the physiochemistry of flotation,² has explained in detail the properties and functions of the various types of reagents used in modern froth-flotation practice. Although his paper is too lengthy to summarize here, it is to be noted that since the publication of this dissertation nothing has been discovered in the art of flotation that his postulates do not explain.

The advancement of the art of modern froth flotation is due in no small measure to the work of numerous investigators in government, university, and private laboratories. Although it is difficult to single out particular individuals for special mention in this connection, the various technical papers by Prof. A. F. Taggart, Prof. A. M. Gaudin, Prof. A. W. Fahnenwald, and O. C. Ralston in the United States and Prof. I. W. Wark in Australia are perhaps outstanding from the standpoint of contributing to a better understanding of the theoretical aspects of flotation.

Types of Ores Amenable to Flotation.—For many years, during the early stages of its development, the flotation process was considered to be applicable to the treatment of sulphide ores only. However, as early as 1905, the possibilities of floating nonsulphide lead and copper ore were explored by Swartz, and U.S. patent 807501 was granted to him covering the use of soluble sulphides to film base-metal oxides and thus assist their flotation. Later work by various investigators, notably N. C. Christensen, led to a fuller recognition of the fundamental differences in the flotation characteristics of sulphides and nonsulphide and nonmetallic minerals. As a result of these studies, techniques and reagents were developed for floating many nonsulphide and nonmetallic minerals. Reagents commonly employed were fatty acids, soaps, and emulsions. The next stage in the development of nonmetallic flotation resulted from the discovery that certain anionic-type wetting agents were more effective than soaps and fatty acids for floating certain types of nonsulphide ores.

More recently, a new class of flotation reagents, of the so-called "cationic" class, has been developed for nonmetallic flotation. These reagents are characterized by

¹ *Trans. Faraday Soc.*, Vol. 15, 1920.

² *Chemistry and the Flotation Process*, American Cyanamid Co., *Tech. Paper* 17, 1930.

the valuable constituents from the gangue by frictional trituration with water. Later on, in 1886, Carrie Everson in the United States disclosed in U.S. patent 348157 the use of fats or oils to buoy sulphides and thus separate them from worthless gangue, the selectivity of the separation being assisted by the addition of acids and acid salts or neutral salts.

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their marked selectivity toward silicate minerals and quartz. By their use, it is possible to effect selective separations that hitherto were impossible to obtain with soaps, fatty acids, or anionic-type wetting agents.

In general, it may be said that, in the light of recent developments in froth flotation, almost all the well-known minerals of commercial importance can be treated by this process. The following is a list of minerals that have been found to be amenable to flotation:

1. Sulphide minerals, such as lead, copper, and zinc sulphides; iron sulphides including pyrite, pyrrhotite, marcasite, and arsenopyrite; cobalt and nickel sulphides; silver sulphides and sulpharsenides and antimonides; arsenic and antimony sulphides; bismuth sulphides; molybdenum sulphide; mercury sulphide; tin sulphides.
2. Metallic minerals, such as copper, gold (also gold tellurides), silver; also metals such as iron, aluminum, and tin.
3. Base-metal oxides and carbonates, including lead carbonate and sulphate; copper oxides and carbonates; and cobalt oxides.
4. Nonbase metal oxides, such as those of iron, tungsten, aluminum, vanadium, manganese, tin, chromium, titanium.
5. Nonbase metal carbonates such as those of calcium, magnesium, manganese, strontium, barium.
6. Sulphates, phosphates, chlorides, fluorides, borates, etc., as for example, barite, gypsum, fluorite, borax, celestite, cerargyrite, cryolite, halite, sylvite, apatite, collophanite, monazite.
7. Coal and carbonaceous materials, sulphur, etc.
8. Silicates, such as talc, feldspar, quartz, garnet, spodumene, mica, kyanite, andalusite, beryl, kaolinite, zircon, nephelite, pyrophyllite, diatomite, sillimanite, vermiculite.

Sizes of Minerals Amenable to Flotation.—Although in some instances, notably in the flotation of coal, it is possible to treat material as coarse as 10 mesh, it may be stated that, in general, froth flotation works best on sizes below 35 mesh. Minerals coarser than this usually are more advantageously treated by other methods of concentration such as tabling, or by agglomerate tabling.

As regards lower limit of sizes, this depends to a large extent on the type of mineral to be floated and the presence of interfering slimes. In general, the floatability of minerals tends to decrease in the range of extreme fineness, in the neighborhood of 600 mesh and finer.

Reagents.—From what has been previously stated in regard to froth flotation, it is readily apparent that the efficiency of the separations obtainable by this process depends on the proper selection of reagents or chemical compounds.

In order to accomplish desired results, a variety of chemical compounds are needed. These may be classified according to their function, as follows:

Frothers.—Production of a persistent froth of desired selectivity and durability is of prime importance in successful flotation. In this regard, it is to be noted that homogeneous liquids do not froth, only aqueous solutions froth appreciably. Frothing takes place when air is bubbled through a liquid if the air-liquid surface tension is capable of a small but measurable rapid change. A good frothing agent must be capable of passing readily into the interface between water and air, so that it must be partly water-repellent and partly water-soluble. In other words, it must be "heteropolar," *i.e.*, possess polar and nonpolar groups.

In this regard, polar water-insoluble saturated hydrocarbons, such as kerosene, give no froth. Unsaturated hydrocarbons give slight froth because they are nonpolar. Organic compounds such as pine oil, alcohols, phenols, fatty acids, and amines give appreciable froths because they contain both polar and nonpolar groups. In this

latter type of compounds, one part of the molecule has an affinity for water and the other has an affinity for air, or conversely, an aversion to water.

From the above considerations, it therefore follows that, in order to be effective, frothers must not be too soluble or too insoluble. In addition, to ensure selectivity in flotation, frothers must not ionize appreciably.

Commonly used frothers in modern flotation practice include pine oil; cresylic acid; higher alcohols; eucalyptus oil; camphor; and the like. A class of reagents known as "froth stiffeners" are used to impart greater stability to the froth in certain instances. Coal-tar creosotes, wood creosotes, and the like, are typical of this class of reagents.

Collectors or Promoters.—Reagents that give an adherence and water repellency to surfaces of minerals to be floated are designated "promoters" or "collectors." These reagents are more or less selective as regards minerals of a certain class. Their selectivity may be enhanced by careful control of the quantity added and by the addition of controlling or modifying agents.

In general, promoters may be considered as falling into the two following classes:

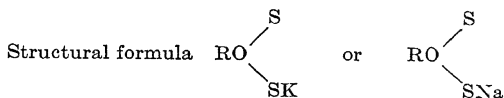
1. Anionic type.
2. Cationic type.

As in the case of frothers, in order that promoters may be effective, they must comprise two parts, one part nonpolar (hydrocarbon) and the other polar, and be capable of adhering by virtue of adsorption or chemical reaction to the surfaces of specified minerals.

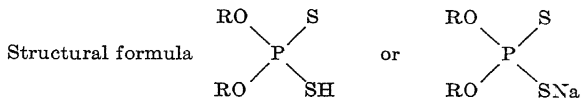
Although resembling frothers to the extent of being heteropolar, promoters differ from frothers in that the polar part of an effective promoter must have a particular affinity for a particular mineral. (The polar part of a frother should have affinity for water only.) Thus, it is found that in the case of promoters for sulphide minerals, for example, the hydroxyl group in the polar part of a frother is replaced by the less water-avid sulphhydryl group.

Anionic-type Promoters.—Commonly used promoters of the anionic-type include

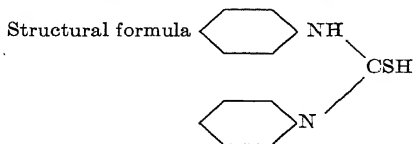
1. Xanthates. The reaction products of an alcohol, sodium or potassium hydroxide, and carbon bisulphide.



2. Aerofloat flotation reagents, or disubstituted dithiophosphoric acid and salts of such acids. These are reaction products of phosphorous pentasulphide and cresylic acid or an alcohol.



3. Thiocarbanilide. The reaction product of aniline and carbon bisulphide.



4. Fatty and aromatic carboxylic acids, such as oleic acid.

Structural formula $R.COOH$

These anionic-type promoters are characterized by the following properties, according to Christmann.¹

1. They are for the most part weak acids. Xanthic acid, for example, is weaker than carbonic acid.
2. They all form alkali-metal salts, at least in solution.
3. They form heavy-metal salts.
4. The free acids are only slightly polar or, in other words, are slightly dissociated.
5. The alkali-metal salts are distinctly polar in comparison.
6. The alkali-metal salts are water-soluble.
7. The heavy-metal salts (Pb, Cu, Ag, Zn) are slightly polar.
8. The heavy-metal salts are quite insoluble in water.
9. The tendency of the heavy-metal salts of all these compounds to dissolve in organic solvents increases as the R in RX of any homologous series increases.
10. In the case of fatty acids, the strength of the acid decreases with the higher members of a series, and the solubility of the acid in water decreases.

In addition to the foregoing list of anionic-type promoters, some less common ones should be mentioned. These are principally used for the flotation of various nonsulphide and nonmetallic minerals.

1. Triethanolamine.
2. Fatty acid residues and their soaps, such as saponified cottonseed oil "foots."
3. Sulphonated fatty acids, glycerides, or oils.
4. Emulsified oils, fatty acids, or glycerides.
5. Sulphated and sulphonated alcohols or "wetting agents."
6. Hydrocarbons such as fuel oil or kerosene. These are sometimes added as vehicles for fatty acids in nonmetallic flotation. For the flotation of minerals such as graphite, talc, or sulphur, they may act as the main promoter.
7. Naphthenic acid, naphthenates, or alkali-metal salts of petroleum oxidation.
8. Carboxylic acids.

The possibilities of utilizing certain reagents of the so-called anionic-wetting-agent class received attention at least as early as 1935. In the initial work with this class of compound, they were tested as emulsifiers for various fatty acids and oils, as well as promoters for certain types of minerals. Much of the original work with these reagents was done by U.S. Bureau of Mines' investigators, and the results were described in two technical publications. The first of these, "New Flotation Reagents" by Dean and Hersberger, was published as *Technical Paper* 605 by the American Institute of Mining and Metallurgical Engineers in 1935. The second article, entitled "Use of Wetting Agents in Flotation" by Dean, Clemmer, and Cooke, was published as *U.S. Bureau of Mines Report of Investigations* 3333 in February, 1937. In this latter publication, Dean and his coworkers made a distinction between two classes of wetting agents, which they represented as being

1. The alkali-metal salts of the higher alcohol sulphates or the long-chain aliphatic acids, which give large *negative* ions in solution.
2. The heavily loaded quarternary ammonium salts, which give large *positive* ions in solution.

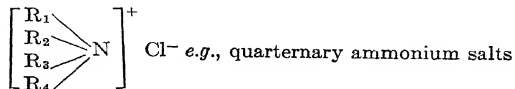
Wetting agents of the anionic type may be represented by the following structural formula:

¹ *Loc. cit.*

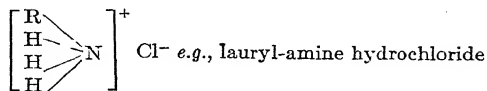
Examples of these types of compounds are the Aerosol wetting agents sold by the American Cyanamid Co.; the Gardinol and Dupanol reagents marketed by E. I. du Pont de Nemours & Co.; certain of the Emulsols marketed by the Emulsol Corp.

Wetting agents of this type find some use as selective promoters for certain non-metallic minerals, also as froth-modifying agents.

Cationic-type Promoters.—Wetting agents of the cationic type may be represented by the following structural formula:



or



Cationic-type wetting agents as a class are characterized by the fact that they are surface-active materials containing both polar and nonpolar groups. In solution, these compounds ionize to give a large positive ion. This positive ion generally has a heavy hydrocarbon loading. These positive-ion type wetting agents may be amines, phosphonium, or sulphonium compounds, but practically all the work that has been done in this country on cationic reagents in ore dressing has been confined to nitrogen compounds.

The point of most interest in cationic-type wetting agents, so far as flotation is concerned, is the fact that such reagents are specific promoters for silica and silicate minerals. Their selective action toward silica and silicates might almost be compared with the action of xanthates or Aerofloat promoters toward sulphide minerals.

The cationic-type wetting agents of the amine type may be primary, secondary, tertiary, or quarternary compounds. These compounds have an amine radical as their effective polar portion and an appropriate hydrocarbon group as their effective nonpolar portion, combined with one another to yield a positively charged surface-active ion in solution. Their potency apparently depends on the ionic grouping and length of the hydrocarbon chain. Experimental evidence indicates that there is a critical chain length of maximum collection for any particular type of compound.

The above-mentioned amine compounds may be either free amines or amine salts, the latter commonly being chlorides, bromides, iodides, or acetates. The various halogen derivatives appear to be about equally effective.

There seems to be some difference of opinion as to whether the cationic wetting agents promote the flotation of siliceous minerals through chemical action or through adsorption. Whatever the theory, the fact remains that certain types of these amine compounds are powerful and selective promoters for quartz and other silicates. The more powerful collectors, *i.e.*, those with long hydrocarbon chain substituents, are used on silicate minerals relatively hard to float or not highly adsorptive. A less efficient reagent with a short-chain substituent works better for talc or clay.

Amine Types.—The various types of commercially available amines that are useful in flotation may be roughly grouped under five classifications, *viz.*,

1. Higher aliphatic amines and amine salts such as lauryl amine and lauryl-amine hydrochloride.
2. Quarternary ammonium salts of the higher aliphatic series, such as cetyl trimethyl ammonium bromide.

3. Pyridinium salts such as lauryl-pyridinium iodide (these are quarternary ammonium compounds of a special type).

4. Alkylated and acylated condensation products of ethanolamines and fatty acids.

5. "Short-chain" amines such as *n*-tributyl amine and triethanolamine.

Addition Agents.—Reagents that are used to modify the normal floatability of minerals with promoters and frothers are designated broadly as "addition agents." This class of reagents may be conveniently subdivided into three general groups:

1. Regulating agents.

2. Activating agents.

3. Depressing agents.

Regulating Agents.—The function of the reagents included in this group is to control the alkalinity of the pulp, also to counteract interfering effects of detrimental slimes, colloids, and soluble salts.

In modern froth flotation, alkaline circuits are used almost exclusively for the treatment of sulphide ores and most nonmetallic ores as well. For any particular ore, there is a definite range of pH at which optimum results are obtainable. Therefore, proper control of this alkalinity is of prime importance. Reagents commonly used to obtain desired alkalinity are lime and soda ash. Lime exerts a flocculating effect, while soda ash acts as a dispersant for ore slimes. Lime is, in most instances, cheaper than soda ash and is more widely used for the treatment of sulphide ores, particularly copper and zinc ores. Soda ash is preferred for the treatment of lead sulphide ores and precious-metal ores, because lime tends to depress galena and free gold and must be more carefully controlled than soda ash.

Besides being an alkalinity regulator, lime is a depressor for pyrite and is widely used for this purpose. If used in excessive quantities, lime also tends to depress many other sulphide minerals.

Caustic soda is occasionally used as an alkalinity regulator.

Sulphuric acid, formerly used to a considerable extent before the advent of modern promoters, is rarely used today. This reagent tends to increase the floatability of all sulphides but is nonselective in its action, besides having the disadvantage of exhibiting a corrosive action on flotation equipment.

Less commonly used than lime and soda ash are certain slime-regulating agents such as sodium silicate, starch, glue, and various phosphates. These reagents are used to aid selectivity by exerting a dispersing action on the slimes and colloids.

Activating Agents.—Reagents of this group are used to assist the flotation of minerals that are normally hard to float with promoters and frothers only. The outstanding example of such function is the use of copper sulphate to float sphalerite and marmatite. Other examples include the use of sodium sulphide to float lead sulphate and carbonate and copper-carbonate minerals; the use of hydrofluoric acid to assist the flotation of feldspar with a cationic-type promoter; the use of lead nitrate for assisting the flotation of various nonmetallic minerals with anionic-type promoters of the fatty-acid type; and aluminum chloride for assisting the flotation of mica with cationic-type promoters.

Depressing Agents.—The function of depressing agents is to assist the selectivity of one mineral from another. Originally, before the development of specific depressants, it was impossible to float minerals of one kind from others whose floatability was closely the same.

The outstanding example of the value of a depressant is the use of alkaline cyanides to separate galena from sphalerite and pyrite, or copper sulphides from pyrite. The discovery by Sheridan and Griswold of the selective depressing qualities of cyanides was one of the most important discoveries in the art of flotation in that it solved the problem of treating complex lead-zinc-iron ores.

Examples of other depressants include the use of

Lime for depressing pyrite.

Sodium and calcium sulphites for depressing pyrite and sphalerite.

Zinc sulphate as an aid to cyanides for depressing sphalerite.

Chromates for depressing galena.

Permanganates for depressing pyrrhotite and pyrite in the presence of arsenopyrite.

Quebracho for depressing calcite when floating fluorite, scheelite, or certain other nonsulphide ores with fatty acids.

Sodium silicate for depressing quartz.

Sulphuric acid for depressing quartz.

Starch and glue for depressing mica and talc.

The "600" series reagents of the American Cyanamid Co. for depressing graphite and carbonaceous slates when floating gold ores.

Caustic soda for depressing ilmenite.

Practical Operation.—The practical adaptation of froth flotation for the treatment of ores involves a sequence of steps, each of which influences the efficiency of the operation as a whole. These various steps may be listed as follows, though not necessarily in the order shown:

1. Liberation of the values by crushing and grinding.

2. Dilution of the ground pulp with water to the desired degree for most efficient flotation.

3. Conditioning of the pulp with reagents prior to flotation.

4. Flotation for separation of desired minerals from worthless minerals.

Liberation of the Values.—Before minerals can be separated by froth flotation they must, of course, be liberated from each other. While, as previously mentioned, flotation can be utilized to treat certain minerals such as coal and phosphate rock as coarse as 10 mesh, in its greatest field of application—the separation of sulphides—the maximum size of the particles rarely exceeds 35 mesh. The lower limit of sizes amenable to flotation is not so well defined and will vary with the type of mineral to be floated, as well as the nature and size of the gangue constituents. In general, sizes in the range of minus 48 mesh to plus 600 mesh respond more readily to flotation. Flotation separations normally are much less efficient on sizes below 600 mesh, and recovery and selection are greatly reduced if the particles are finer than 5 to 10 microns.

The ore fed to flotation is almost universally wet ground in a ball mill or rod mill operating in closed circuit with a classifier. In grinding the ore for subsequent treatment by flotation, an effort is made to produce the minimum amount possible of extreme fines, or slimes. This desirable condition is accomplished by using short large-diameter ball mills in closed circuit with classifiers capable of handling large circulating loads. In some operations, avoidance of slime and savings in costs are accomplished by a procedure whereby the ore is ground only to the point where a rejectable flotation product, or "tailing," can be obtained. The low-grade froth, or "concentrate," consisting of incompletely liberated sulphides or other mineral values in gangue, is reground until the unlocking is complete and then refloats for production of a finished concentrate.

As previously stated, oversliming is to be avoided, since slimes interfere with the separation of the coarser minerals and, in addition, cause higher consumption of reagents.

Dilution of the Ground Pulp.—For effective separation by froth flotation, it is necessary that the ground pulp be sufficiently dilute to give the minerals ready mobility. Generally, in the primary, or "roughing," stage of flotation, the pulp contains a

higher percentage of solids than in the cleaning stage wherein the rougher concentrate is retreated to produce the final high-grade product.

In the treatment of most ores, the roughing operation is performed at pulp densities ranging from 20 to 35 per cent solids. Occasionally, when the ore contains a large amount of minerals of high specific gravity, pulp densities as high as 40 to 60 per cent solids may be used, but this is uncommon. Slimy ores may require the rougher flotation to be conducted at 10 to 15 per cent solids.

Pulp densities in cleaning operations commonly run from as low as 3 to as high as 18 per cent solids and occasionally higher.

Comparative testing will establish the optimum pulp density for conducting roughing and cleaning operations for any given ore. In general, both speed of flotation and selectivity are improved in dilute pulps, but the larger volume to be handled will usually require more flotation-machine capacity. Reagent consumptions are also usually greater when dilute pulps are handled.

Conditioning of the Ore Pulp Prior to Flotation.—Although many of the reactions between reagents and minerals in the flotation process are practically instantaneous, there are instances where appreciable time is required to bring about the desired effects. In such cases, the pulp must be "conditioned." This is usually accomplished by providing a tank, of suitable capacity and equipped with means for agitating the pulp, between the grinding circuit and the flotation machine.

The conditioning step usually follows the grinding step. However, in some instances the ground ore may require some special treatment prior to conditioning and flotation. Some refractory types of ores may require the removal of extreme fines or colloids in order to improve the efficiency of the subsequent flotation operation. In some instances, improved results are obtained if the ground and deslimed ore is "scrubbed" in an attrition-type mill, sometimes in the presence of special cleansing agents, to remove adhering surface coatings and stains, and is again deslimed.

In many instances, improved results are obtained if the conditioning step is conducted at a high pulp density—sometimes as high as 70 to 80 per cent solids. Reduced reagent consumptions usually follow such a procedure, and the volume of conditioning equipment required is also less than when treating more dilute pulps. However, high solids conditioning usually entails the use of some dewatering device or thickener between the grinding circuit and the flotation-machine circuit. Sometimes, air is also introduced into the conditioning tank to assist in the desired reactions.

Occasionally, the pulp temperature is raised during the conditioning step, in order to speed up the reactions between minerals and reagents and thus aid selectivity and recovery. In such cases, the usual method of heating is by introduction of live steam through a pipe or pipes inserted into the pulp.

Reagents added in the conditioning stage may be regulators such as lime or soda ash (although the usual practice is to add these particular reagents to the grinding mill); activators such as copper sulphate or sodium sulphide; or depressants such as zinc sulphate, alkaline cyanides, alkaline sulphites, and sodium silicate.

Frequently, promoters or collecting reagents, such as Aerofloat flotation agents or xanthates, are also conditioned with the pulp prior to flotation. In most instances, it is desirable to add such reagents subsequent to the regulating, activating, or depressing agents. This involves the use of several conditioning tanks in series.

As previously mentioned, it is common practice to add to the grinding mill certain of the reagents required for the flotation operation. In such case, the grinding circuit serves the same purpose as a conditioning tank. Alkalinity regulating agents and depressants such as lime, soda ash, sodium silicate, sodium cyanide, and zinc sulphate are usually added at this point. Frequently, promoter reagents are also added to the grinding mill in addition to the alkalinity regulators. However, when depressants

are added to the grinding circuit, it is best to add the promoter at a later stage in the operation.

Flotation Operation.—In the actual flotation operation, the separation of the minerals is accomplished in specially designed machines. A considerable variety of such flotation machines are in use. In general, they fall into three classes, based on the manner in which air is introduced into the pulp, as follows:

1. Mechanically agitated, no outside air added. In this type, air is drawn in by suction through the action of the rotor or impeller. Examples of machines of this type are the Fagergren (Fig. 1); Denver Equipment Co.; Weinig.

2. Mechanically agitated, with addition of outside air. In this type, the pulp receives some air through the action of the impeller and also from air introduced under low pressure from a blower. Examples of machines of this type are the Minerals Separation subaerated machine and the Booth-Thompson Agitair machine.

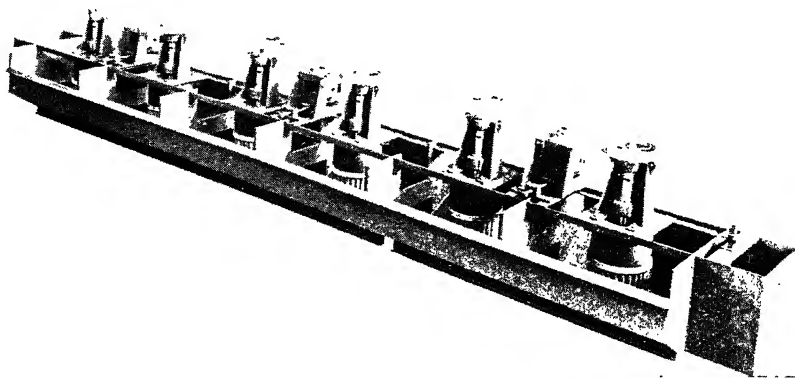


FIG. 1.—Fagergren flotation unit (latest type).

3. Pneumatic-type machines. In this type, the air is directly introduced into the cell at low pressure from a blower. Examples of this type are the Steffensen air-flotation machine (Fig. 2), the Southwestern Matless air-lift machine, the MacIntosh machine, the Callow pneumatic machine, and the Britannia deep air-lift machine.

Regardless of type, flotation machines must include suitable means for introducing pulp, maintaining it in suspension, and regulating the rate of discharge of the unfloatable portion of it. At the same time, all machines must provide means for introducing and dispersing air into the pulp and means for discharging the froth, as for example, by the use of mechanically operated skimmers, and conveying it to the next step in the operation.

Flotation machines are usually composed of several identical cells arranged in series, the unfloatable portion of the pulp flowing from the discharge opening in one cell to the intake opening of the succeeding cell.

In large operations, parallel rows of a number of machines in series must be used. It is good practice to provide a relatively large number of cells for roughing operations, to avoid dangers of short-circuiting of the pulp. In cleaning operations, a smaller number of cells may be used, inasmuch as the tailing from such operations is not a finished product but is returned to the original feed going to the rougher machines.

Choice of flotation machine is dependent on many factors in addition to metallurgical and mechanical efficiency. However, the modern trend is toward machines of the straight mechanical type capable of providing intense agitation and a large volume of finely dispersed air into the pulp.

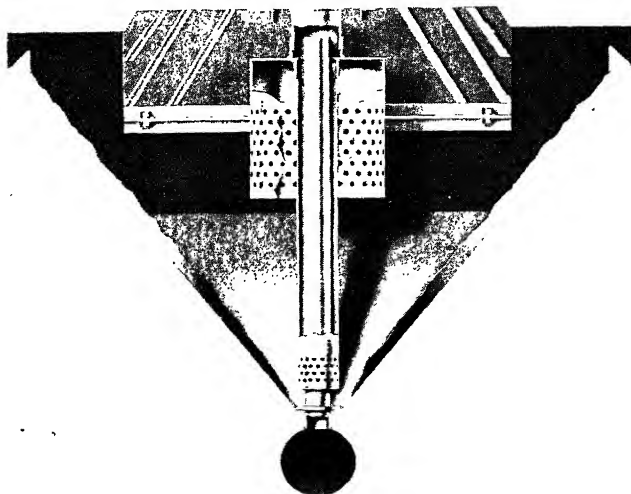


FIG. 2.—Steffensen air-flotation machine (air introduced from header underneath apparatus).

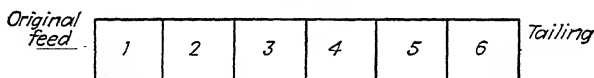


FIG. 3.—Flotation flow sheet (single roughing flow).

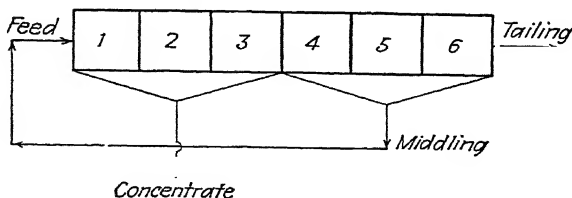


FIG. 4.—Simple concentrate-middling flow scheme.

Next in importance to the proper selection of reagents for the flotation operation is the feeding of these in controlled, uniform amounts to the grinding circuit, the conditioners, or to the flotation machines. Requirements for efficient-type feeders

include precision and reliability, ease and range of adjustment, resistance to attack by the reagents fed, simplicity of design, and low cost of operation and maintenance.

Solid reagents, such as lime and soda ash, are usually fed by means of slow-moving belt conveyers, electrically vibrated conveyers, a horizontally rotating plate and cutter, or a screw conveyer operating in a trough.

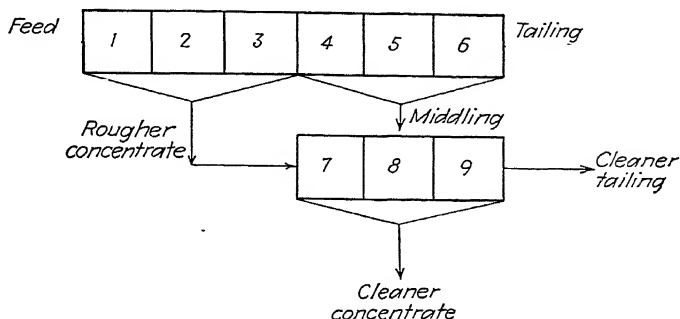


FIG. 5.—Rougher-middling circuit with rougher-concentrate cleaning.

*Original
feed*

*Cleaned
concentrate*

Tailing

FIG. 6.—Roughing and cleaning operations in a single circuit.

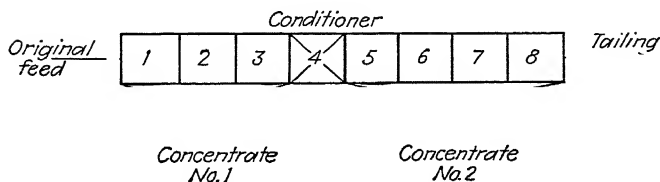


FIG. 7.—Differential flotation using one cell (No. 4) as conditioner.

Many types of feeders are available for dispensing liquid reagents and aqueous solutions. Commonly used types include pulley and finger, disk and cup, and reciprocating cup. The disk-and-cup type is in more general use than any other liquid-reagent feeder. In this type a vertical disk is submerged in the liquid and rotates on a horizontal axis. A number of small cups or buckets are suspended on short rods attached to the disk in such manner that they may be tipped a certain predetermined amount so that their contents may be discharged into a collecting

trough leading into pipes that convey the reagent to the desired point in the flotation circuit. An example of this type of feeder, the Clarkson, is illustrated in Fig. 9.

Reciprocating-cup feeders work on the same principle as the self-dumping mine skip. The feeder consists essentially of a large reservoir into which a beaker-shaped

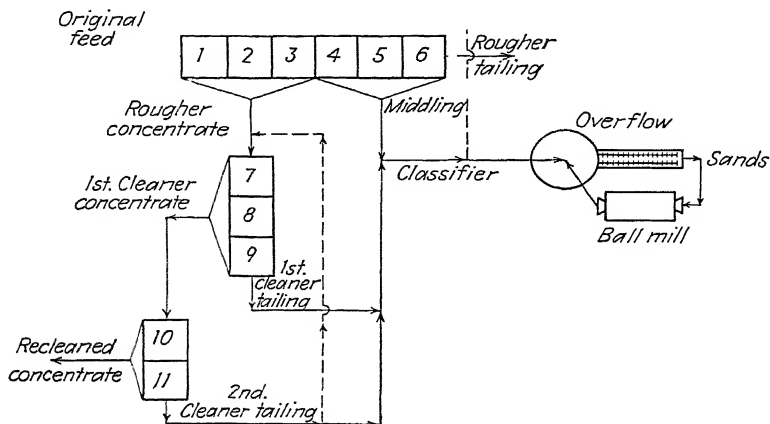


FIG. 8.—Rougher-double-cleaning circuit with regrinding.

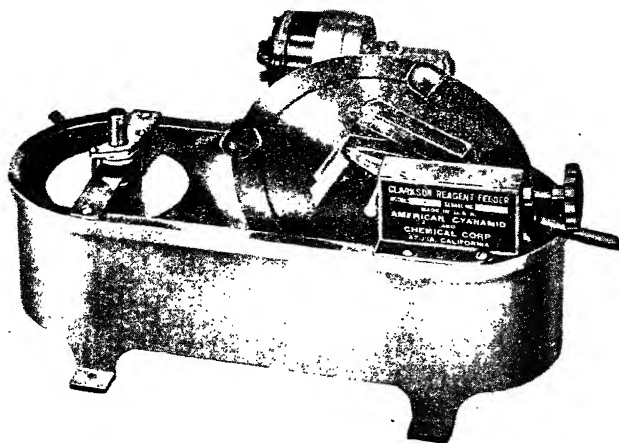


FIG. 9.—Clarkson reagent feeder.

cup is lowered and raised by means of a suitable reciprocating device. The rate of feed is controlled by adjusting the angle to which the cup is tipped at the point of discharge. This type of feeder is especially adapted to the feeding of large quantities of liquid reagents as in certain nonmetallic flotation operations.

APPROXIMATE QUANTITIES AND METHOD OF FEEDING THE COMMON REAGENTS USED
IN FLOTATION

Reagent	Usual quantities fed		Usual method of feeding	Recommended point of addition
	Lb per ton	G. per metric ton		
Promoters:				
Aerofloat 15.....	05- 0.20	25- 100	Undiluted	Conditioner or grind- ing circuit
25.....	05- 0.20	25- 100	Undiluted	
31.....	05- 0.20	25- 100	Undiluted	
239.....	02- 0.10	10- 50	5-10% soln.	
241.....	02- 0.15	10- 75	5-10% soln.	
242.....	02- 0.15	10- 75	5-10% soln.	Conditioner, grinding circuit or flotation circuit
203.....	05- 0.20	25- 100	5-10% soln.	
208.....	01- 1.10	5- 50	5-10% soln.	
213.....	02- 0.10	10- 50	5-10% soln.	
226.....	02- 0.10	10- 50	5-10% soln.	
238.....	02- 0.10	10- 50	5-10% soln.	
243.....	02- 0.15	10- 75	5-10% soln.	
Sodium Aerofloat.....	05- 0.20	25- 100	5-10% soln.	Conditioner or grind- ing circuit
Sodium Aerofloat B...	05- 0.20	25- 100	5-10% soln.	
Reagent 404.....	20- 0.50 ^a	100- 250	5% soln.	
425.....	20- 0.50 ^a	100- 250	5% soln.	
444.....	20- 0.50 ^a	100- 250	5% soln.	
Ethyl xanthate.....	05- 0.15	25- 75	10% soln.	Conditioner grinding circuit or flotation circuit
Butyl xanthate.....	05- 0.15	25- 75	10% soln.	
Amyl xanthate.....	05- 0.15	25- 75	10% soln.	
Reagent 301.....	05- 0.15	25- 75	10% soln.	
343.....	05- 0.15	25- 75	10% soln.	Grinding circuit Grinding circuit or conditioner or flota- tion circuit
Thiocarbamilide.....	05- 0.10	25-	Dry	
Thiocarbamilide 125...	05- 0.10	25-	5-10% soln.	
Fatty acids.....	50- 2.	250-1000	{ Undiluted or dissolved in fuel oil }	Conditioner; also added in stages dur- ing flotation
Reagent 708.....	50- 2.			
Frothers:				
Pine oil.....	05- 0.20	25- 100	Undiluted	Flotation circuit
Cresylic acid.....	05- 0.20	25- 100	Undiluted	
Frother 52.....	05- 0.20	25- 100	Undiluted	
Frother 60.....	05- 0.20	25- 100	Undiluted	
Coal tar oils.....	05- 0.20	25- 100	Undiluted	
Coal tar creosotes....	05- 0.20	25- 100	Undiluted	
Aerofloat 15.....	05- 0.20	25- 100	Undiluted	
Reagent 712.....	05- 0.20	25- 100	5-10% soln.	

APPROXIMATE QUANTITIES AND METHOD OF FEEDING THE COMMON REAGENTS USED IN FLOTATION.—(Continued)

Reagent	Usual quantities fed		Usual method of feeding	Recommended point of addition
	Lb per ton	G. per metric ton		
Activators:				
Copper sulphate.....	0.5 - 2.0	250-1000	Saturated solution	Conditioner
Sulphuric acid.....	0.5 - 2.0	250-2500	Undiluted	Conditioner
Sulphidizing agents:				
Sodium or barium sulphide.....	0.5 - 5.0	250-2500	10% soln.	Conditioner or flotation circuit
Depressants:				
Cyanide.....	0.03- 0.75	15- 375	5-10% soln.	Conditioner or grinding circuit
Dichromate.....	1.0 - 5.0	500-2500	5-10% soln.	
Sodium sulphite.....	0.10- 5.0	50- 500	5-10% soln.	
Zinc sulphate.....	0.10- 2.0	50-1000	5-10% soln.	
Starch.....	0.01- 0.20	5- 100	Caustic starch, 5-10% soln.	
Lime.....	1.0 -10.0	500-5000	Dry or as milk of lime	Conditioner or grinding circuit
Sodium silicate.....	0.5 - 3.0	250-1500	10% soln.	
Modifying agents:				
Lime.....	1.0 -10.0	500-5000	Dry or as milk of lime	
Soda ash.....	1.0 - 5.0	500-2500	Dry or as 10% soln.	
Sodium hydroxide....	0.5 - 3.0	250-1500	10% soln.	Conditioner or grinding circuit
Sodium silicate.....	0.5 - 3.0	250-1500	10% soln.	
Sodium sulphide.....	0.5 - 3.0	250-1500	10% soln.	
Sulphuric acid.....	1.0 - 5.0	500-2500	Undiluted	
Ammo Phos.....	0.5 - 3.0	250-1500	Dry	
"600 Series" reagents	0.10- 1.0	50- 450	2-5% soln.	Conditioner or flotation circuit

^a For base-metal oxide flotation.^b For nonmetallic flotation.*Reagent Formula*

The following is a useful formula that may be used to calibrate the amount of reagent being fed to flotation:

$$\text{Pounds per ton} = \frac{\text{cubic centimeters per minute} \times \text{specific gravity of liquid}}{0.315 \times \text{short tons per 24 hr.}}$$

or

$$\text{Pounds per ton} = \frac{\text{grams per minute}}{0.315 \times \text{short tons per 24 hr.}}$$

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CHAPTER VII

DEWATERING

BY JOHN V. N. DORR¹ AND FRANK L. BOSQUET²

Definition.—Dewatering, as used in metallurgical work, is the separation of a mixture of solids and water into two parts, one of which is relatively solid-free, and the other relatively liquid-free, with respect to the original mixture.

The various methods of accomplishing dewatering include (1) draining; (2) sedimentation, or thickening; (3) filtration thickening; (4) filtration; (5) drying. The last usually requires the application of artificial heat and is covered in an earlier chapter.

Draining can be applied effectively only to the coarser sand fraction of a pulp, and it consists merely of allowing the water to run off the mixture under the action of gravity.

Sedimentation, or thickening, involves the settling of the solid fraction out of a pulp suspension under more or less quiescent conditions. Material ranging from about 35 mesh to particles in the colloidal range (if sufficiently flocculated) can be dewatered by this means.

Filtration thickening involves a combination of these two methods of removing water and, in practice, is carried out generally in a machine resembling a thickener except that it is also provided with filtering elements.

Filtration is the process of mechanically removing undissolved particles from a liquid by passing the liquid through a permeable medium, such as paper, cloth, or sand. Filters are of two general classes—pressure filters and vacuum filters—depending on which method of forcing the liquid through the filtering medium is applied.

Various combinations of these methods are employed. Frequently coarse sands and slimes are dewatered separately. Usually dewatering by sedimentation (thickening) is employed ahead of filtration, the latter being more costly and functioning better on a relatively thick feed.

DRAINING—STATIONARY

Draining may be regarded as a crude form of filtration. A simple example is a pile of wet sand on a floor; most of the liquid moves down through the interstices between the grains and eventually runs off, only that held by adhesion and capillarity remaining. The filling of railroad cars with wet traprock from screening plants or with coarse jig tailings, where the excess water runs off through openings in the cars, is a typical case of dewatering by stationary drainage.

Watertight bins may be used instead of floors. In this case the sand-water mixture is fed into the bin, and the water, which is continuously replaced by the sand, is allowed to overflow the top as the bin fills up. When full, the stream is diverted and the bin dug out. Sometimes "filter panels," made up of screen cloth, are employed for drainage in bins, these panels running vertically along the inside walls of the tank.

However, the use of such methods is largely obsolete in modern metallurgical practice. The disadvantages are obvious—the time and labor required are excessive and the tanks have a low capacity, considering the actual time the tank is working.

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Acknowledgment is here made of the assistance of the staff of The Dorr Co.

In order to permit better drainage and also to provide a means of percolating solution through the sand, tanks equipped with a filter bottom are used in cyaniding in some places where the sand and the slime are separately treated. Vats of 50 ft. in diameter with a capacity of 750 tons are used in the older plants on the Rand. The filter at the bottom consists of wooden gratings, made in sections to permit convenience in handling, on which is laid a covering of jute matting, and on that a second mat of coconut fiber. In place of the matting, canvas can be used. Later practice in sand treatment on the Rand was to superimpose one vat on another, the upper functioning as a collecting vat and the lower as a treatment vat.

For filling the vats, Butters & Mein or Blaisdell distributors are used. The B & M distributor consists of a number of radial pipes with flat nozzles at the end, rotating about an axis and distributing the pulp evenly in the tank from a hopper. The force of the pulp leaving the arms is sufficient to rotate the mechanism. The Blaisdell distributor consists of a hopper feeding the pulp onto a rapidly revolving horizontal steel disk, which, aided by radial vanes, spreads the pulp evenly and lightly in the tank.

For emptying intermittent tanks, the settled solids are shoveled out, sluiced out of the bottom, or removed by some excavating device such as the Blaisdell excavator, which is merely a set of revolving plows that rake the material to the center of the tank. The machine is gradually lowered into the tank and continually moves the solids toward a central discharge.

SEDIMENTATION

Mechanical Dewatering.—The sandy portion of a classified product from a mechanical classifier of such types as the Dorr and the Akins is dewatered by the draining and mechanical squeezing action of moving the sand up an inclined slope, and in many cases the dewatering thus effected is sufficient to meet requirements. A classifier produces a cleaner sand than any cone system. Mechanical classifiers are extensively used to dewater concentrates and tailings. When dewatering concentrates, the sand discharge from a mechanical classifier will run from 8 to 15 per cent moisture, depending upon fineness and gravity.

Mechanical rake classifiers may be equipped with a pusher plate on the last flight of rakes and a steeply inclined extension on the discharge lip to effect a squeezing action on the sand product just before discharge. Frequently, a 3 to 4 per cent reduction in moisture content is obtainable in this way, particularly in the case of coarse granular solids.

The Dorrco sand washer, a special type of mechanical classifier generally used in concrete sand washing, gives a satisfactory and inexpensive removal of free moisture from granular sand products. It consists of a circular tank containing a slowly revolving wheel equipped with buckets that drag the sand up a peripheral deck inclined at 4 in. per ft., and discharge them into a chute. Excess water overflows a lip at the lower end of the tank. The 12 ft. diameter unit has a capacity of 100 to 150 tons of sand per hour.

Principles of Sedimentation.—In general, sedimentation may be defined as a "movement of solid particles through a fluid, due to an imposed force which may be gravitational, centrifugal or some other force."¹

A particle suspended in a liquid settles because of the unbalanced force represented by its weight opposed to the resistance of the water. The rate at which the particles in a pulp settle governs the capacity of a settling tank. This rate of settlement

¹ STEWART and ROBERTS, *Trans. A. I. Ch. E.*, Vol. 11, p. 124, 1933.

depends on many factors, varying with the physical and chemical characteristics of the solid material, with the presence of electrolytes in the water, with the temperature, and with the density of the pulp.

All the characteristics listed under Classification govern the rate of settlement of particles, although colloidal behavior is encountered throughout the various phases in the thickening of a typical pulp.

Particle size is of great importance, for when particles are so small as to be in the range of semicolloids, or true colloids (finer than 0.0001 mm.), settling may be materially retarded or stopped altogether. It is obvious that the total surface exposed by these particles is enormous compared with their actual mass. In a dispersed pulp, each particle carries an electric charge and these repel each other, setting up an intricate zigzag motion known as "Brownian movement," which is sufficient to keep them dispersed despite the pull of gravity. To cause the colloid particles to settle, it is necessary to neutralize the charge on the particles, when the solids cohere and become in effect a coarse suspension. This phenomenon is termed "flocculation."

Flocculation may be effected in several ways—by the addition of an electrolyte, the introduction of a colloid of opposite sign, or a combination of the two. Electric currents and biological action are both used, but their commercial application in metallurgy is not extensive. The electrolytes most commonly used are lime, aluminum sulphate, alum, ferrous sulphate, and sulphuric acid.

Colloids having an opposite charge to those of the pulp may be introduced either by chemical precipitation or by direct introduction. Lime and alum or lime and ferrous sulphate will form gelatinous precipitates of aluminum or ferrous hydrate which contain relatively high positive charges when in the colloidal condition, which is attained at the instant of precipitation. The use of caustic starch is an example of a colloidal flocculating agent.

When both electrolyte and colloid are used, the former flocculates the colloid added, the flocs formed serve as nuclei to which the original particles adhere, and the whole, settling, tends to collect all suspended material by occlusion or entrainment of the finer material. The subject of colloids and their treatment is a branch of physical chemistry, a further discussion of which is beyond the scope of this book.¹

The rate of settlement depends also upon the temperature—a rise of temperature decreasing the viscosity of the liquid medium, and thereby increasing the settling rate of the solids. The capacity of a settling tank may vary as much as 20 per cent between winter and summer. In introducing heat, precaution should be taken against the setting up of convection currents within the tank due to unequal heat distribution, for in extreme cases this can interfere seriously with settlement.

It is generally accepted that most settleable suspensions separate into sludge and clear liquid in several stages, best explained by means of the cylinders shown in Fig. 1. *E* shows a suspension as fed to a thickener, thoroughly mixed. Heavy particles, if present, will settle immediately, as shown by the dark bottom layer in the following cylinders. The rest of the pulp soon shows four more or less distinct layers or zones. Zone *B* of the original consistency appears to fall away from the surface, leaving the clear liquid *A*. Zone *D* is a layer of particles which, having settled, rest directly one upon the other, with liquid still in the interstitial spaces. Zone *C* is a transitional layer, the consistency of which varies from that of *B* to that of *D*. Obviously, zones *B* and *C* will grow gradually smaller and finally disappear, leaving *A* and *D*, separated by a line called the "critical point." This marks the end of free settling and the start of settling in the compression zone; *i.e.*, the particles, resting on one another, can go no further by mere settling, and any increase in density must come by the upward

¹ The reader is referred to a paper by E. J. Roberts, Colloidal Chemistry of Pulp Thickening, *Trans. A.I.M.E.*, Vol. 112, p. 178, 1934.

passage of water through spaces or channels in *D*. This final step governs the percentage of moisture in the final sludge.

There are, in general, two principal steps involved in sedimentation: clarification, constituting the first stage, and thickening, the second. In the very dilute suspensions encountered in the first stage, the clarification rate is dependent on the falling velocity of the slowest settling flocs, but in the thickening zone proper there exists a condition that might be termed "collective particle subsidence." Here the rate of settlement approximates that of the average falling velocity of the individual particles in a medium whose density depends at each moment on the dilution of the water-solid mixture.

As the compression zone is reached, the settling rate is greatly retarded owing to the fact that the exudation of water can now take place only by floc consolidation and channeling. After the water between the flocs is eliminated, the water tends to be squeezed out of the flocs themselves by the weight of the overlying particles. An equilibrium between the forces involved is eventually reached, and further dewatering can then be accomplished only by (1) mechanical stirring, such as by the action of

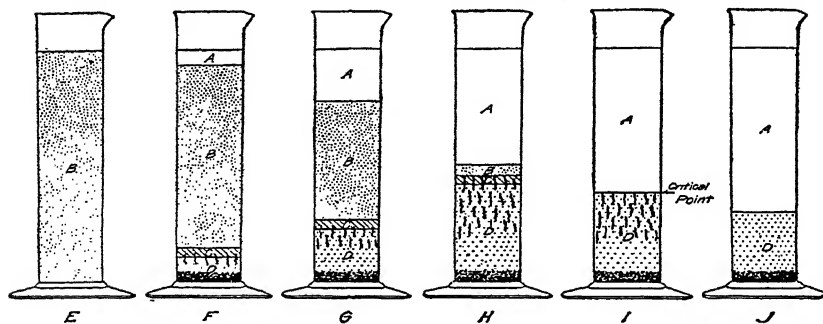


Fig. 1.—Stages of slime settling. (After Coe and Clevenger.)

rakes in a thickener, which tends to rupture the flocs and rearrange the constituent particles, or (2) by the application of greater exuding force, such as by filtration.

Development of Thickening Devices.—The necessity of thickening the dilute slime pulp overflowed from the various types of apparatus used to separate the sand from the slime has led to great improvements. This need was particularly urgent in the concentrators built to handle the porphyry copper ores in the western United States during the years from 1905 to 1915. In these mills, it was necessary to thicken the dilute slime overflow from classifiers to a consistency suitable for vanner feed. With the later trend toward flotation and the "all-slime" practice in the treatment of gold ores by cyanidation, the necessity for efficient apparatus was further emphasized. Today in practically every field of ore treatment, the dewatering of fine material by settling or thickening is practiced to some extent.

Prior to the latter part of the last century, the only method of thickening ore slimes was an intermittent one in settling tanks. The objections to batch thickening are obvious, and numerous attempts were made to make such operations continuous. The first definite advance was made by fitting the intermittent tanks with peripheral overflow launders and running the dilute pulp continuously to a central feed well as long as a clean overflow could be obtained. After this, the solids in suspension were allowed to settle, the supernatant liquor was decanted, and the settled sludge was removed.

The next development was the introduction of cone settlers, which aimed to give not only a continuous clear overflow, but also a continuous or semicontinuous underflow of thickened sludge. Large numbers of these cones were installed in a wide range of operations, notwithstanding the difficulties caused by the building up of solids on the sides, the lack of uniformity of underflow, and the close operating attention required. Furthermore, settling is primarily a function of area, and the size to which cones could be built was limited by practicability of construction. This was a great disadvantage, as a multiplicity of cones was required for an operation of any magnitude. The Magna mill of the Utah Copper Co., before alterations, contained 264 cones 7 ft. in diameter.

J. V. N. Dorr, in remodeling a milling plant in the Black Hills, conceived the idea of operating a mechanism or scraper to move slowly the settled solids to a discharge or underflow opening in the bottom of a cylindrical tank and still not interfere with settling. From this idea developed the Dorr continuous thickener, which in various improved types is now used in virtually all hydrometallurgical and ore-dressing work and has made settling continuous while affording a continuous underflow of sludge of uniform density and quantity.

This machine also made possible the economical and efficient washing of fine solids by the method known as "continuous countercurrent decantation," which is today the accepted and widely used method of washing finely ground particles free from liquids containing dissolved substances.

Cones.—Conical tanks such as the Merrill, Caldecott, and Callow (Fig. 2) were at one time widely used to thicken slime and fine sand, particularly in connection with table concentration, but are in little use today owing to their limited capacity. The

Callow cone consists of an inverted cone with sides sloping at an angle of 60 deg. The cone illustrated is 8 ft. 9 in. in diameter at the top and 7 ft. 8½ in. deep. The feed enters through a launder to a feed well in the center, not shown in the figure. The thickened pulp settles to the bottom of the cone, and the clear water overflows into the peripheral launder. The pulp is continuously discharged through the gooseneck, which is made in three sections. The lower section is made of 1½-in. pipe and is provided with a plug cock to admit water or compressed air to clean out the cone or gooseneck. The second section consists of 1½-in. rubber hose, and the third a piece of 1½-in. pipe curved as shown in the figure. The consistency of the pulp is regulated by raising or lowering the point of discharge, its distance below the water level ranging between about 12 and 24 in.

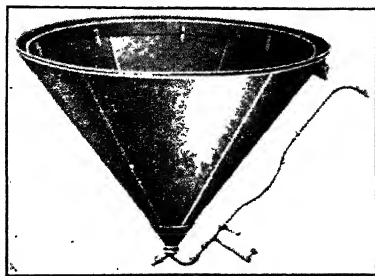


FIG. 2.—Callow cone.

An inspection of Table 1 will show that cones have a small capacity and are generally suited to small operations only. When used in batteries, they occupy more space than a single tank of equal total settling area and capacity, and there is the added annoyance and expense of the manual attention required.

The Homestake mill has a dewatering system partly employing Merrill cones. The slime-thickening division employs conical-bottomed tanks, the largest of which are 32 ft. in diameter. A cone of this size is 24 ft. high, the top 3 ft. being vertical and the bottom sloping at an angle of about 45 deg. A central feed and annular launder for the overflow are provided. The discharge is from a casting at the bottom of the cone, passing into a horizontal pipe leading to a vertical siphon discharge, the

final outlet being a small cast-iron replaceable bushing feeding into a launder. The thick mud accumulating on the sloping bottom does not flow down with absolute uniformity, a certain proportion piling up and sliding down at irregular intervals. With a battery of several such cones, the total overflow varies only slightly and little operating attention is required. High-pressure water connections and shutoff valves permit washing in case of choking.

Table 1 gives the approximate capacities of Callow cones in gallons per minute.

TABLE 1.—CAPACITY OF CALLOW CONES

Size, ft.	When used as sloughing-off tank, gal. per min.	When used as settling tank, gal. per min.
2	24	
2½	40	
3	56	
4	100	6- 8
5	160	10-12
6	230	14-18
8	400	25-30

On the Rand, cones were at one time extensively used to dewater the slime, but these have been largely replaced by thickeners. Cones as large as 50 ft. in diameter have been used and are, in general, similar to the Merrill cones described above.

The *Allen cone* is an improved type in which there is an automatic control over the discharge of settled solids. When the collected solids in the cone reach a certain depth, a "float" is caused to rise, opening the discharge valve through a system of levers. The Allen cone is illustrated in the preceding chapter.

Thickeners.—Several makes of continuous thickeners involving the mechanical plowing of the settled material to a discharge opening or openings are used in metallurgical plants. The principal ones are the Dorr, the Hardinge, and the Genter, the latter being a combination thickener filter.

The *Dorr thickener* is used for sedimentation dewatering of metallurgical, chemical, sanitary, and industrial pulps of all kinds. It consists essentially of a circular tank or basin in which is located a slowly revolving mechanism with a central vertical shaft carrying one or more sets of arms provided with plow blades that sweep the settled solids to a central discharge opening. The feed enters continuously around the periphery of the tank.

The mechanism is supported by either a central pier or by structural members spanning the tank. A lifting device, manual or motorized, together with an overload alarm system, prevents undue strains on the mechanism due to abnormal feed conditions.

The rotation of the radial arms carrying the plow blades is not sufficient to stir up the contents of the tank. The slight gentle movement given to the pulp has proved generally to be of distinct benefit in obtaining a thickened discharge by opening passages for entrained water to rise to the surface.

There are a number of types of Dorr thickeners: the type A central shaft, the torq, and the traction, for operation in the single-compartment tanks; the balanced type and washing type which operate in multiple-compartment tanks, horizontally subdivided by one or more trays.

Sizes range from 6 to 325 ft. in diameter for single-compartment units, and from 10 to 90 ft. for tray units. There may be two to seven compartments in the latter.

Standard construction is of iron and steel for the mechanism, and steel, wood, or concrete for the tank. In the case of some of the larger single-compartment units, earth-fill tank construction is employed. For the handling of corrosive substances, special construction involving rubber covering, stainless steels, wood, and lead covering are employed.

Type A Central Shaft.—This central-shaft Dorr thickener, in which the mechanism is supported from a bridge "superstructure" spanning the tank, is shown in Fig. 3. It is normally used in single-compartment units up to 40 ft. in diameter. The thick-

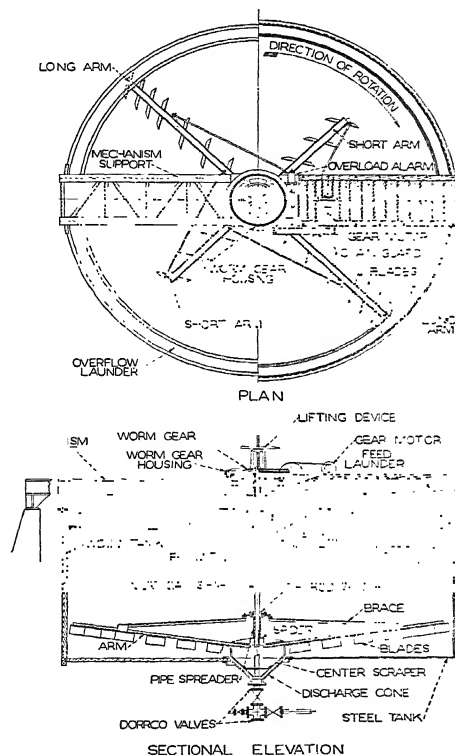


FIG. 3.—Central-shaft Dorr thickener.

ener mechanism consists of a vertical central shaft carried by bearings and a supporting bracket on the superstructure. The large worm gear is supported around its periphery by a ball bearing, giving the shaft rigidity. The arms are fastened to a spider at the lower end of the shaft. The rake arms are inclined to the horizontal plane, with the result that a slightly conical bottom of settled solids is built up under the plow blades. A discharge cone at the bottom receives the solids delivered by the rakes, and from the cone the pulp is usually withdrawn by a diaphragm pump.

The Dorr Torq Thickener.—This is a single-compartment thickener in which the mechanism is supported by a column or pier at the center of the tank, thus avoiding

the need for a superstructure spanning the top of the tank. These units have been built up to 250 ft. in diameter and are becoming standard for general metallurgical work.

This machine gives automatic protection against overloads in that the arms are hinged at the central rotating structure in such a way that they swing upward when an overload is encountered and return to their normal position when such overload condition has been passed. Either two or four radial arms are employed, depending on the raking requirements.

The name "torq" is derived from the unique torque lifting action of the arms. Figure 4 shows how the arms function under "normal" and "overload" conditions.

Feed is introduced either through a central inverted siphon within the central column or through a radial launder or pipe, supported by a light bridge-type overhead truss. Overflow is collected in a peripheral launder, and sludge is raked into the center where it is deposited in either an annular trough or a central cone, and withdrawn by pump.

The Dorr Traction Thickener.—The mechanism for this thickener is designed especially for large-diameter installations and the handling of large-tonnage tailings

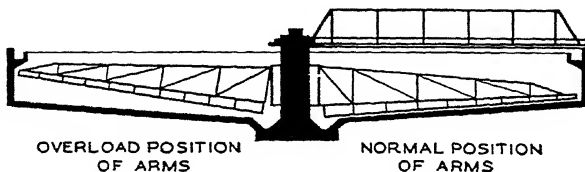


Fig. 4.—Dorr torq thickener.

dewatering jobs. Its outstanding feature is the support of the raking truss from a rail mounted on top of the tank wall. Figure 5 shows a 200-ft. diameter unit at the plant of the Utah Copper Co. The driving power is applied at the rail end of the raking arm by means of a motor-driven carriage. The inboard end of the slowly revolving truss rests upon a turntable mounted on a central stationary pier. Rake blades are secured to the lower chord of the truss and to one or more shorter stub arms providing additional raking capacity near the center. A walkway gives access to the central turntable for inspection and lubrication. Feed is carried to a central feed well by means of a pipe or launder supported by a truss. Overflow is collected peripherally, and sludge is discharged from the central sludge well by means of sludge pipes that are carried through a tunnel or set in the bottom of the tank.

Traction-type units are generally furnished in sizes from 200 to 325 ft. in diameter. Structural ruggedness and tangential application of power permit its use in the size range beyond that of the torq type.

Multiple-compartment Thickeners.—The Dorr balanced-tray thickener, as shown in Fig. 6, consists of two or more superimposed settling compartments operating in "parallel." There is a raking mechanism in each compartment, but all are driven by means of one central shaft from a single drive mechanism, mounted on a superstructure spanning the top of the tank.

The feed pulp is divided between the compartments in a feed-splitter box. Overflows are taken from each compartment separately, but are combined in a common collection box. The combined sludge from the various compartments is collected at the bottom of the tank and is discharged by a Dorco diaphragm pump.

Each compartment is effectively sealed off from the others by a sludge seal consisting of a downcast cylinder or boot attached to the central opening of the tray and

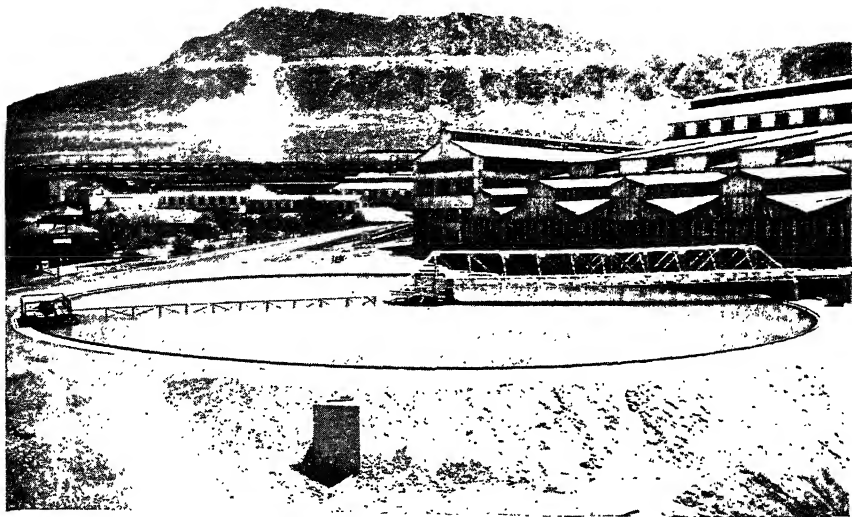


FIG. 5.—Traction thickener (200-ft. diameter).

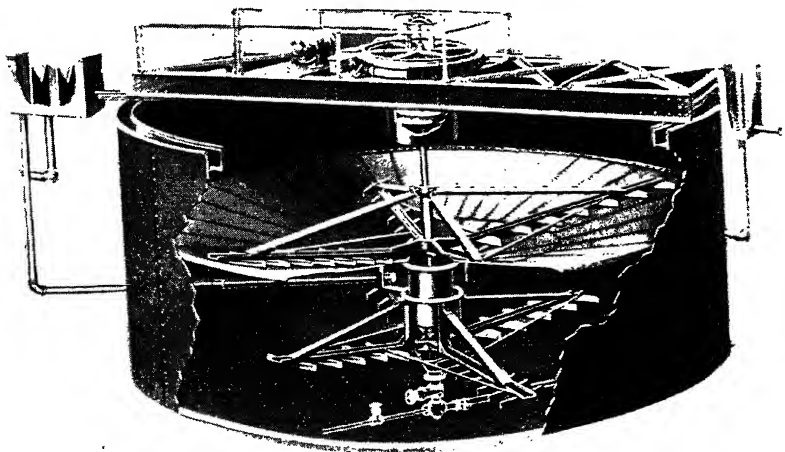


FIG. 6.—Dorr balanced-tray thickener.

extending to within a short distance of the next lower settling surface. This provides a continuous set of conduits for the flow of thickened sludge to the common outlet. Short-circuiting is thereby eliminated, and full tray efficiency is obtained.

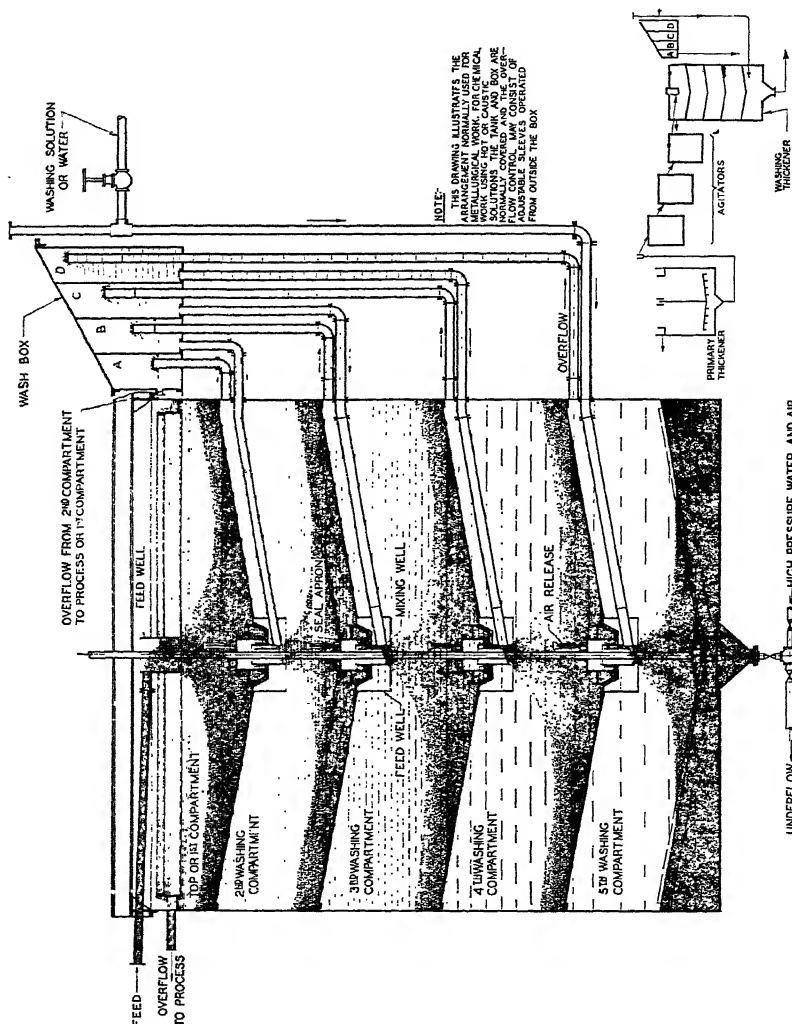


Fig. 7.—Dorr washing-tray thickener.

The sludge blanket depths in the various compartments are maintained and can be controlled by adjusting the height of the overflow standpipes. Each compartment acts essentially as a separate thickener. The balanced-type thickener gives

maximum capacity per unit of ground area. Its normal size range is 10 to 80 ft. in diameter, with two to six compartments.

The Dorr washing-tray thickener is similar in general construction to the balanced type, but the multiple compartments in the washing-type thickener operate in series.

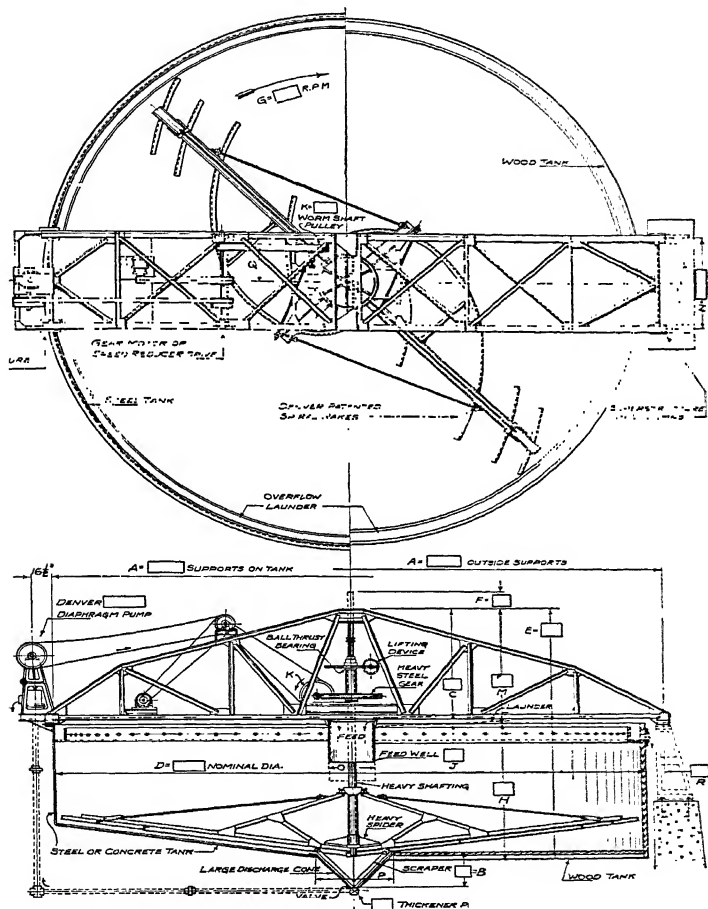


FIG. 8.—Denver Equipment Co. thickener (bridge type).

This allows countercurrent washing of a sludge as would be obtained in a series of unit thickeners, but at a large saving in space.

Figure 7 shows how the washing-type unit functions. The interconnecting piping and the sludge-seal arrangement between compartments are quite different from those in the balanced type. The top compartment receives all the new feed and overflows

all the liquid. The lower compartment receives the wash water and discharges all the washed solids. Washing takes place in each compartment in C.C.D. series.

Dorr washing-tray thickeners are normally built in sizes from 10 to 60 ft. in diameter and with two to six compartments. Special units are built where the top com-

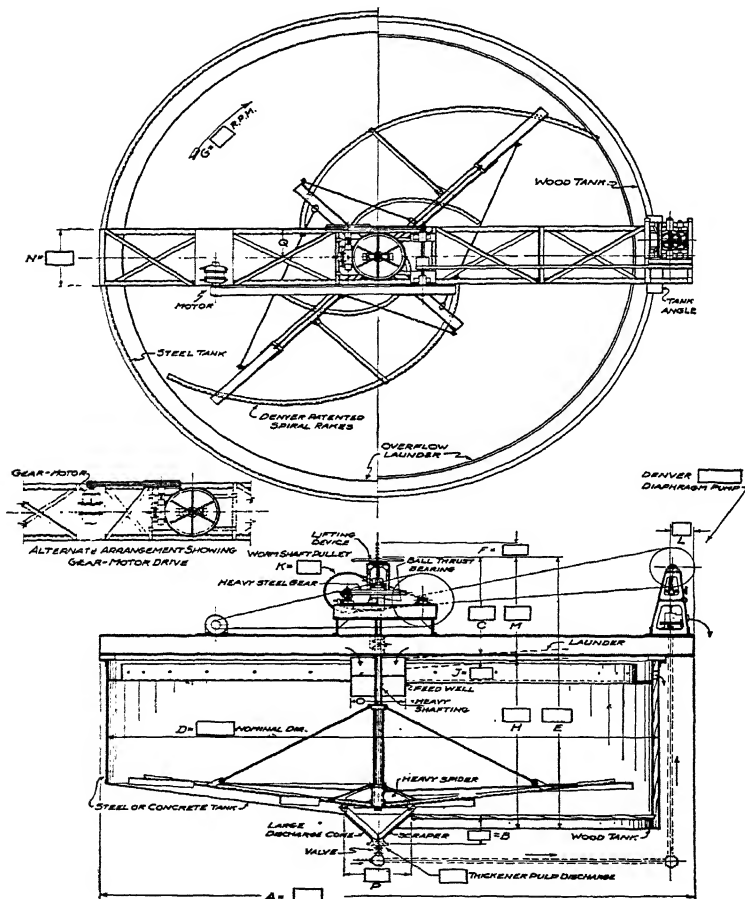


FIG. 8a.—Denver thickener (beam type).

partment is sealed off from those below and is equipped with a separate underflow discharge controlled by a diaphragm sludge pump. This provides one separate stage of primary thickening with several stages of countercurrent washing of the suspended solids. Where metallurgical practice calls for agitation of pulp between primary thickening and C.C.D. washing, this arrangement is particularly applicable.

Denver Thickener.—The Denver thickener, made by the Denver Equipment Co., in single-compartment and tray types, employs either straight blades or spiral rakes. Its mechanism is of the central-shaft type.

Superstructures of the "beam" type are used in sizes up to 45 ft. diameter, and bridge or "B" superstructures are employed in sizes up to 75 ft. diameter. Figures 8 and 8a show both units.

Hardinge Thickener.—The Hardinge thickener raking mechanism consists of a continuous spiral, as illustrated in Fig. 9. This is a central-shaft single-compartment

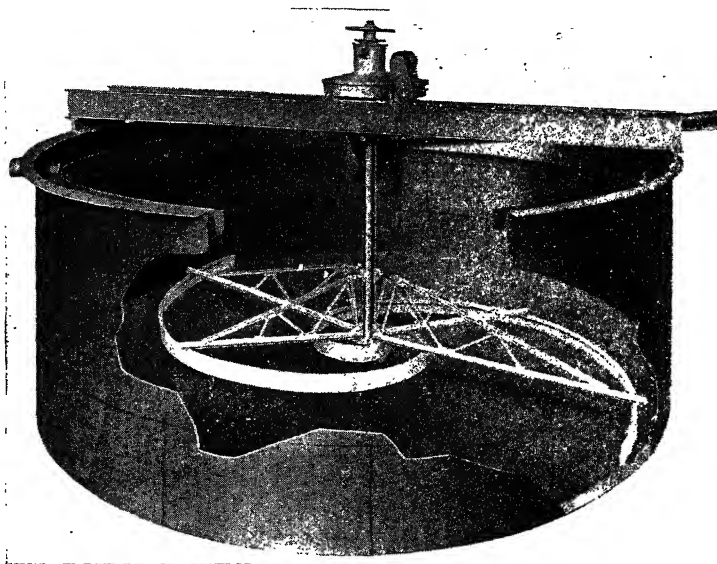


FIG. 9.—Hardinge thickener.

thickener, and the drive and drive-support structures are generally similar to those of the Dorr and Denver central-shaft units.

FILTRATION THICKENING

Genter Thickener.—The Genter thickener is a combination thickener filter, employing hollow filter tubes suspended in a settling tank. Suction and pressure are alternately exerted upon the tubes. As a result, a cake of solids is formed on the outside of the tubes and is released when the pressure is reversed. The solids fall into the mixture being thickened, and eventually settle into a cone-shaped bottom with a discharge opening at its base, to which the solids are moved by a slowly revolving rake.

Figure 10 shows a sketch of the machine which consists of two concentric tanks, the inner one acting as the vacuum receiver. The annular space between the two tanks is the thickening chamber, which is divided into eight compartments by walls that radiate from the inner tank. A number of filter tubes are suspended in each compartment, and each group is connected to a rotating filter valve, which periodically

produces a vacuum alternating with a pressure blowback within the tubes. This type of thickener was developed for use in the treatment of first carbonation juice in beet-sugar factories, for the separation of the sugar juices from lime sludge. It has, however, found some application in metallurgical work. In the handling of flotation

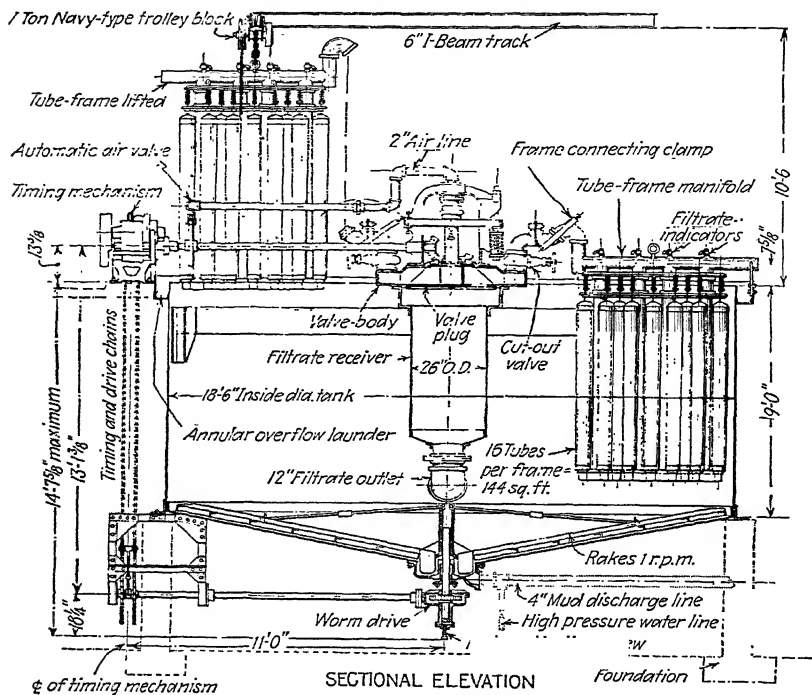


FIG. 10.—Genter thickener.

concentrates, the capacity of these machines runs in the range approximately of 150 to 350 tons solids per 1000 sq. ft. per 24 hr.

TABLE 2.—SIZES AND FILTRATION AREAS OF GENTER FILTER THICKENERS¹

Tank diameter	Filter area, sq. ft.	Tank diameter	Filter area, sq. ft.
2' 2"	50	13' 0"	1150
9' 0"	375	14' 4"	1300
12' 0"	750	18' 6"	2300
12' 6"	940		

¹ From General Engineering Company "Metallurgical Handbook."

The Oliver-Borden Thickener.—In this filter thickener the rectangular tank is divided by a vertical plate into two hopper-bottomed compartments. Impeller

mixers in the V bottoms repulp the discharged cake before removal through a goose-neck pipe outlet. Perforated steel tubes, wrapped spirally with wire and covered with filter cloth, are suspended vertically in the tank. An automatic valve mechanism applies alternately vacuum and pressure to the tubes through a header.

The Sweetland Filter Thickener.—This is similar to the Oliver-Borden except that the filter tubes are fluted wood cylinders instead of steel tubes. There is also a somewhat different mechanical arrangement for providing the alternating vacuum and pressure.

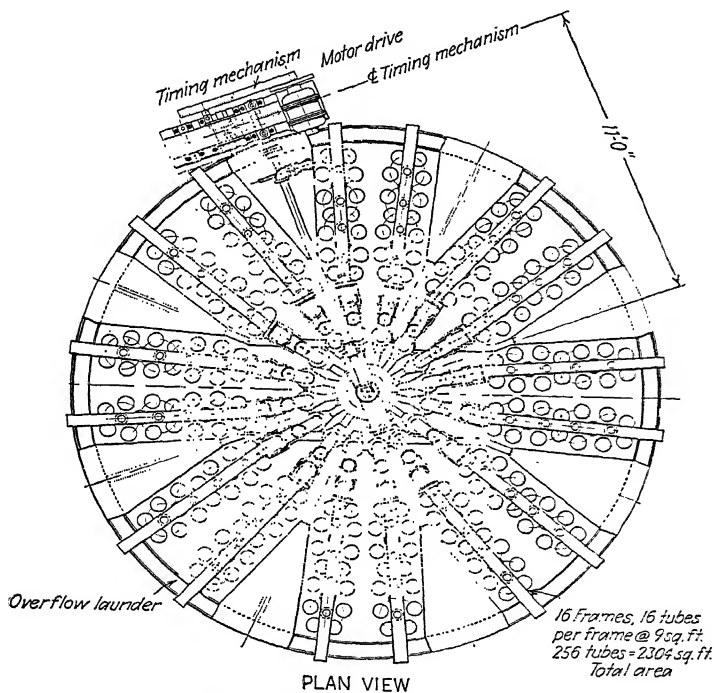


Fig. 10a.—Plan of Genter thickener.

The Shriver Filter Thickener.—This is a modification of the Shriver plate and frame filter press except that the frames are so designed that the cake does not build up on the plates. The solids are continuously swept out of the unit as a slurry, separately from a stream of clear filtrate.

Continuous Counter-current Decantation.—This process, or C.C.D., as it is commonly termed, is carried out with a series of thickeners operating in such a manner that the solids with a small portion of liquid pass successively from one thickener to the next, being finally discharged from the last thickener as washed tailings. The main body of liquid, on the other hand, moves in the opposite direction to the course of the ore and is gradually enriched. In each step of the system, a large amount of

low-grade solution is mixed with a very small amount of higher grade solution, so that the value of the solutions entering each successive thickener is materially reduced.

A study of the flow sheet (Fig. 11) will illustrate the principles of washing by countercurrent decantation. The grinding equipment, thickener 1, and the agitators are not shown. The following assumptions are made: 100 tons of ore per day are ground in cyanide solution, and \$10 per ton in gold is dissolved; the pulp is discharged from each thickener in the ratio of one of liquid to one of solids by weight, or 50 per cent moisture. The dotted lines represent the flow of thick pulp containing all the solids, and the solid lines, the flow of clear solution. The figures followed by T indicate in tons per day the flow of solution with assay values as shown.

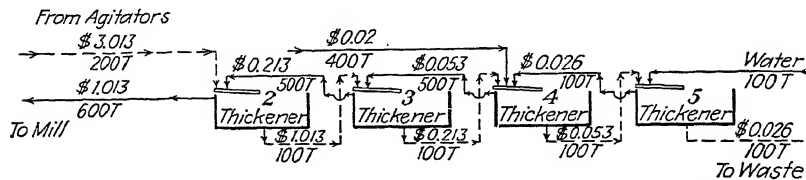


Fig. 11.—Flow sheet—washing by continuous countercurrent decantation.

It is assumed that dissolution of the gold is completed in the agitators. The pulp solution after agitation, having a value of \$3.013 per ton, is shown entering thickener 2 at a dilution of 2:1. It is here diluted with 500 tons of solution from thickener 3 having a value of 21.3 cents per ton. The resultant mixture has a value of \$1.013. Six hundred tons are overflowed from thickener 2 and are returned to the grinding circuit and agitators. The underflow, containing 100 tons of solution, is sent to thickener 3, where it is diluted with 500 tons of solution having a value of 5.3 cents contained in the overflow from thickener 4. The resultant mixture contains 600 tons of solution, with a value of 21.3 cents, and this comprises the feed to thickener 3.

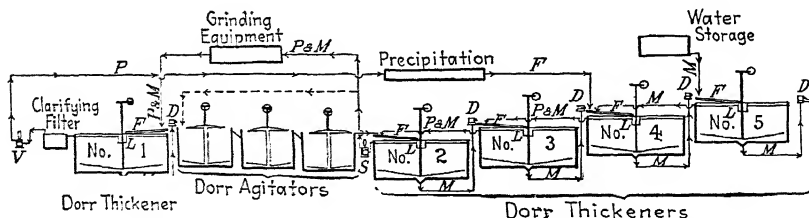


Fig. 12.—Typical flow sheet of continuous countercurrent decantation.

The underflow from thickener 3 will then be 100 tons with a value of 21.3 cents per ton. This underflow is mixed with 400 tons of barren solution having a value of 2 cents per ton, and 100 tons of overflow from thickener 5, which has a value of 2.6 cents per ton. The resultant mixture of these three volumes is the feed to thickener 4, which is 600 tons with a value of 5.3 cents per ton. The underflow from thickener 4 of 100 tons of solution at 5.3 cents per ton is diluted with 100 tons of water, having no value, this mixture comprising the feed to thickener 5. The underflow from thickener 5 of 100 tons of solution at a value of 2.6 cents per ton and 100 tons of tailings is discharged to waste. The water added to the last thickener is equal to the amount lost in the tailing, thus maintaining the balance of solution in the system. The barren solution is added at thickener 4 instead of at 5 in order to avoid a high mechanical

TABLE 3.—EQUATING C.C.D. CYANIDE SYSTEM
(REFER TO FIG. 10)

Thickener 2

In

200 tons at	\$3.013	=	\$602.60
500 tons at	0.213	=	106.50
			<u>\$709.10</u>

Out

600 tons at	1.013	=	\$607.80
100 tons at	1.013	=	101.30
			<u>\$709.10</u>

Thickener 3

In

100 tons at	1.013	=	\$101.30
500 tons at	0.053	=	26.50
			<u>\$127.80</u>

Out

500 tons at	0.213	=	\$106.50
100 tons at	0.213	=	21.30
			<u>\$127.80</u>

Thickener 4

In

100 tons at	0.213	=	\$ 21.30
400 tons at	0.02	=	8.00
100 tons at	0.026	=	2.60
			<u>\$ 31.90</u>

Out

500 tons at	0.053	=	\$ 26.50
100 tons at	0.053	=	5.30
			<u>\$ 31.80</u>

Thickener 5

In

100 tons at	0.053	=	\$ 5.30
100 tons at	0.000	=	0.00
			<u>\$ 5.30</u>

Out

100 tons at	0.026	=	\$ 2.60
100 tons at	0.026	=	2.60
			<u>\$ 5.20</u>

loss of cyanide, though a slightly lower loss of dissolved value would be obtained by putting it in the latter. Table 3 summarizes the C.C.D. metallurgical balance.

A typical arrangement of the apparatus employed is shown in Fig. 12. The step system of elevation of the thickeners is shown; *i.e.*, each succeeding thickener in the decanting series is stepped up above the preceding one, in order that the clear-solution overflow may be transferred by gravity. On the flow sheet, *D* represents the pumps used for the transfer of the thickener underflow, *F* the feed or mixing launders in which suitable baffles or riffles are placed to accomplish a thorough mixture of the pulp entering the thickeners, and *L* the loading or feed wells, which receive and introduce the feed a short distance below the surface of the solution. *V* represents the pump that supplies the necessary suction to the clarifying filter and also elevates the

chemical industries. A flow sheet illustrating its use in a copper-leaching plant is shown in Fig. 13. In this case a Dorr multideck classifier is used to leach and wash the sandy material, and thickeners are used for washing the slimes.

Diaphragm Pumps. *Dorrco "VM" Pumps.*—It is very important in most thickening operations, particularly in C.C.D. operations, to control the volume and density of discharge from any thickener. The Dorco "VM" diaphragm pump (Fig. 14), was developed for this purpose. This pump is of the constant volume-displacement type and employs cord rubber diaphragms and ball valves. Several sizes from $1\frac{1}{2}$ up

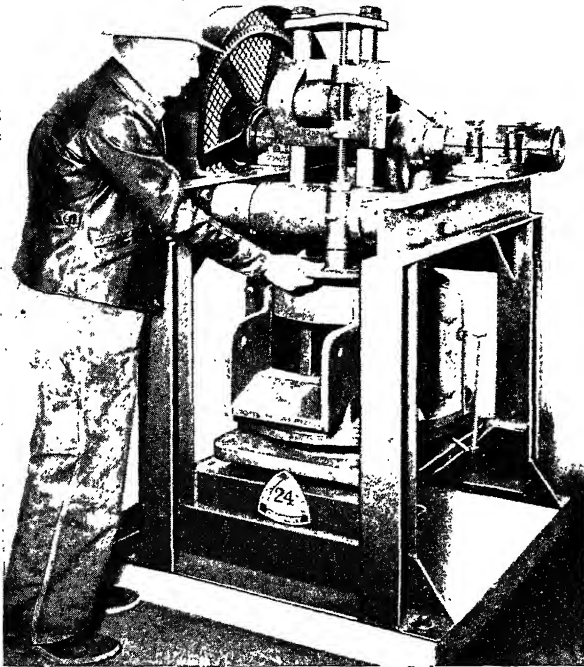


FIG. 15.—Denver Equipment Co.'s diaphragm pump.

to 6 in. are built to meet tonnage requirements. All are of the duplex type. Capacity can be closely regulated by means of a stroke adjustment that can be made while the pump is in operation.

The Dorco pressure pump, which is also of the diaphragm type, but with a special diaphragm arrangement, makes it possible to pump against a discharge pressure of about 20 lb. per sq. in.

Denver Pump.—The Denver Equipment Co.'s diaphragm pump and those of several other manufacturers are similar in general arrangements to the Dorco and are used for the same purposes. Figure 15 shows the Denver motor-driven simplex pump. These pumps are made in sizes from 1 to 6 in.

FILTRATION¹

By DONALD F. IRVIN

Theory.—The fundamental principle upon which slime filtration depends is that if a filter medium of some permeable material such as canvas is immersed in a liquid containing solids in suspension, and a difference in pressure is created between the opposite sides of this medium, the liquid portion of the mixture will pass through the medium leaving the solid portion on the surface in the form of a cake.

The fundamental requirement of a difference in pressure may obviously be applied in one of two ways—by a pressure greater than atmospheric on the feed side, or less than atmospheric on the outlet side, and the method used determines whether the filter is of the pressure or vacuum type. If a pulp is placed in a canvas bag and pressure is applied at the neck of the bag, the liquid will obviously go through and the solids be retained by the canvas. This, then, would be the simplest type of chamber filter or filter press. But if, instead of forcing the liquid from the inside out, the pulp is placed in an airtight vessel surrounding the bag, the bag being supported on a frame and communicating with the atmosphere, and pressure is applied to the surrounding vessel, the liquid will pass through to the inside of the bag and the solids will adhere to the outside. This is a pressure-leaf filter. If, instead of putting pressure on the outside of the bag, the inside were connected to a vacuum pump, the result would be the same as before, but, as another means of obtaining the difference in pressure is utilized, this would be a new type, known as the suction-leaf filter. In order to refine the apparatus as much as possible, the bag would be placed on a circular frame and suspended horizontally in the pulp. If the inner surface of the canvas is connected to a vacuum pump and the apparatus rotated slowly, the solids will continually form a cake on the surface of the canvas, which can be scraped or blown off as the drum revolves. This is known as the "rotary vacuum filter" and is a widely used type.

It is obvious that with a vacuum filter the difference in pressure cannot exceed 14.7 lb. per sq. in., while with a pressure filter the only limit is the strength of the machine and the economy involved.

The capacity of a filter, expressed in either the amount of cake formed or the amount of filtrate produced, is directly proportional to the time required for the operating cycle. Expressed more directly, capacity is the volume either of the filtrate or of the cake, per unit area, per unit of time, or

$$C = \frac{V}{AT}$$

where C = capacity.

V = unit of volume.

A = unit of area.

T = unit of time.

But since $V = AL$, where L = thickness of cake, then

$$C = \frac{AL}{Am}$$

which is simply an expression of velocity or rate of flow. It can, therefore, be stated that the capacity of a filter is directly proportional to the rate of flow of the liquid

¹ Oliver United Filters, Inc., New York.

through it, and any formula developed for rate of flow can be translated to a capacity statement. Efforts have been made to derive a fundamental filtration formula, but it has been exceedingly difficult to make one formula applicable to all conditions.

Sperry¹ has developed a formula that appears to be satisfactory for filtration problems where the sludge is of a homogeneous nature, such as sand or calcium carbonate. He assumes that filtration consists of two processes: the flow of liquid through a porous mass, and the building up of a cake. For the first process, the flow of liquid through a porous mass, Sperry gives the formula

$$\frac{dQ}{dT} = \frac{P}{RT + R_m}$$

where dQ/dT = rate of flow of liquid with respect to time.

P = difference in pressure.

R = resistance of cake.

R_m = resistance of filter base.

T = thickness of material.

For the second process, the building up of a cake, he derives the following formula:

$$t = \frac{\%Q}{K}$$

where t = thickness of cake.

$\%$ = per cent of solids in mixture.

Q = total quantity of discharge.

K = rate of deposition.

Combining the two formulas and solving for Q , the total quantity of discharge, give Sperry's fundamental formula

$$\sqrt{\frac{2PKT}{R\%} + \left(\frac{KR_m}{R\%}\right)^2} = \frac{KR}{R}$$

Where a temperature other than standard is used, the whole expression is multiplied by N_s/N_o , where N_s equals the coefficient of viscosity of the liquid under standard conditions of temperature and N_o is the coefficient of viscosity at the temperature used.

Sperry notes that two other factors should be introduced, one a factor modifying P , the pressure, to allow for the squeezing together of nonrigid solids as the pressure increases; and another to take care of the influence of gravity through the agency of sedimentation in the filter.

F. P. Baker² shows that the pressure should be modified by an exponent n to compensate for the compressibility of the sludge. He proposes the following formula as the fundamental law of filtration, essentially the same as that of Lewis and Almy,³ derived some years before:

$$\frac{dV}{dQ} = \frac{KA^3P^n}{V^m}$$

where dV/dQ = rate of flow, in volume per unit of time.

K = proportionality constant, dependent on nature of sludge.

A = area of filtering surface in square feet.

P = difference in pressure in pounds per square foot.

V = volume of filtrate in cubic feet.

n = constant, dependent on nature of sludge.

m = constant, dependent on nature of sludge (varies from 1.8 to 2.2).

¹ *Met. Chem. Eng.*, Vol. 15, p. 198.

² *Jour. Ind. Eng. Chem.* Vol. 13, p. 610.

³ *Jour. Ind. Eng. Chem.*, Vol. 4, p. 528.

To apply this formula it is necessary to determine the constants K , n , and m by experiment for any particular sludge, and from them the size of installation required can be calculated. The constant n will be greater than unity where there are two solids in suspension, one being of a porous nature, and less than unity for any single solid in suspension. For noncompressible homogeneous solids, n will be unity or will approach it very closely. It is true that Baker's formula does not express separately the resistance of the cake, the percentage of solids in the mixture, and the rate of deposition, but these are all determined experimentally and come under the constant K . Baker's formula, however, involves an expression area which Sperry's does not, facilitating the translation of data directly to the calculation of sizes of filter equipment.¹

Professor George J. Young² made an extensive study of slime filtration, and his experiments have resulted in the standardization of a large amount of information on filtration. He summarizes the subject of slime filtration as follows: (1) The proportion of clayey material in ores that are to be filtered should be kept at a minimum. (2) The slime pulp should be as free as possible from plus 150-mesh sand and should contain as much as possible of minus 200-mesh material. (3) The slime pulp, prior to filtration, should be thickened as much as possible. (4) The temperature of the slime pulp should be maintained between 20 and 30°C., or even higher. (5) The temperature of the wash water and that of the pulp should be alike. (6) Vacuum pressures should be varied until the proper intensity for the given slime is obtained. (7) If very clayey slime is to be filtered, as much fine sand (limited as in 2) should be crowded into the pulp as it will carry without undue settling and clogging. (8) Number 10 canvas supported by slats gives the best all-round service for the thick-cake machines and No. 12 canvas for the thin-cake machines. (9) With slime containing a large proportion of colloid or clayey material, pressures greater than those obtainable with vacuum apparatus are of questionable advantage. (10) With slime containing a large proportion of clayey material, the vacuum filter should be used. (11) With slime containing a small proportion of clayey material and much fine sand, both vacuum filters and pressure filters could be used with perhaps equally good results. (12) With slime containing much coarse and fine sand, chamber filters with air agitation and high pressure would perhaps give the best results. (NOTE:—The rotary sand filter would also seem to be applicable.) (13) Of the vacuum filters, the thin-cake continuous filters are a decided improvement over the thick-cake filters.

Alliot³ has studied the subject of washing in filter presses and has determined that the following factors prevent the displacement from being perfect: (1) *Adsorption*, or the unequal distribution of the solubles at the boundary between solid and liquid phases of the cake, owing to surface-energy effects. A large excess of solubles is retained in the surface film and will not wash away. (2) *Absorption* of mother liquor by *capillary diffusion* into the pores of the cake particles requires a dilute wash to be effective, and a time factor comes into effect. (3) *Chemical combination* with the substance of the filter particles has little effect in metallurgical processes, but often comes into play in the manufacture of chemicals. (4) *Colloid formation* from the removal of electrolytes which hold the particles in the coagulated state may enter into the process. As the electrolyte is washed away, a finer structure results, causing a reduction in the rate of flow.

¹ From a practical viewpoint, it is possible to establish expected filter performance by a series of tests using familiar and recognized test methods. To evaluate the various constants in filtration formulas also requires thorough testing, so that the former method remains the generally favored basis to fix filter rates.

² *Trans. A.I.M.E.*, Vol. 42, p. 783.

³ *Jour. Ind. Eng. Chem.* Vol. 13, p. 976.

In obtaining the best results for thorough washing in a filter press, it appears necessary to have a plate-and-frame press, a means for air elimination, and the wash must leave at the top of the press so that the cakes remain immersed in the bath of wash, eliminating unbalanced hydrostatic head.

In vacuum filtration, the quantity of wash solution required to eliminate the mother liquor will be approximately two and three times the quantity of mother liquor remaining in the cake.

Even when using this amount, it is not practicable to remove the mother liquor completely, as the wash water will go through any cracks that develop, or if the cake is not homogeneous it will pass through the more porous portion without displacing the liquor in the remainder.

History.—Filtration, in its present state, is a highly developed art, but this development is comparatively recent and has been closely allied with the improvements in the cyanide process. The industrial world today is indebted to the gold metallurgists who, in solving their own particular slime dewatering problems, gave to the world the machines and knowledge they developed.

Filtration probably had its inception in the bag filter, the familiar article used for making jelly in the kitchen. The filter press was invented in England about a century ago and, though modern practice has shown a tendency to more continuous apparatus, the continued use of the filter press speaks for its efficiency.

As the cyanide process developed, engineers realized that the old plate-and-frame presses were unsuited to their operations, because of their low capacity and high cost of operation. George Moore, as superintendent of a cyanide mill, invented his suction leaf filter which solved his problem and undoubtedly went a long way toward the present-day efficiency of the cyanide process. Another suction-leaf filter was invented by Charles Butters and came into wide use. Its adoption led to protracted patent litigation, and while the fight was at its hottest the advantages of continuous counter-current decantation became widely recognized and a majority of plants abandoned Moore and Butters filters in its favor.

As vacuum filtration is limited to a difference in pressure of 15 lb. per sq. in., several machines have been developed utilizing pressure instead of vacuum, overcoming at the same time the difficulties encountered in filter-press operation. David J. Kelly, an associate of Moore, developed the first of these machines, his particular problem being filtration in a plant at a high altitude where a vacuum installation was under a severe handicap. Kelly's machine retained all the advantages of suction-leaf filters besides providing several new ones.

Ernest J. Sweetland, a cyanide-mill superintendent, developed his well-known "clam-shell filter," operating on the Kelly principle, but differing in design. This machine has been modified considerably in the last ten years and is today one of the leaders of the pressure filters.

In the course of his work on vacuum filters, Moore evolved the idea of the rotary vacuum filter, and before his company went out of business he turned the idea over to the International Filtration Co., which improved upon the design and put the machine on the market as the Zenith filter.

At about the same time, E. L. Oliver invented his rotary vacuum filter, which is today entitled to the position of leader of the rotary vacuum filters because of its extended use. It is made by the Oliver United Filters, Inc.

The American continuous vacuum filter uses rotating disks instead of a drum, making large filter areas per unit of floor space possible.

The American disk, as well as the Kelly and Sweetland pressure, filters are now made by the Oliver United Filters, Inc.

Filter Presses.—The filter press as exemplified by the Shriver and Sperry machines consists of a framework on which rests a series of plates covered with filter cloth and spaced a small distance apart, so that when the plates are clamped together between a fixed and movable head they form a series of chambers, walled in by filter cloth. There are two means of spacing the plates, and the method used denotes whether the machine is of the plate-and-frame or of the recessed-plate type.

In the operation of either type, the cloth is first laid over the plates, centering the holes over the conduit openings in the press. The press is then closed and the pulp pumped in. The pressure is gradually raised to the maximum, beyond which efficient filtration ceases. Wash water is introduced if required. To dewater the cake completely, compressed air or steam is usually introduced to replace the water. The locking mechanism is then released and the movable head slid back, the frames being tipped or jerked to dislodge the cake, which falls to a conveyer or to pans beneath the press. The plates and frames are then reassembled for the next cycle of operations.

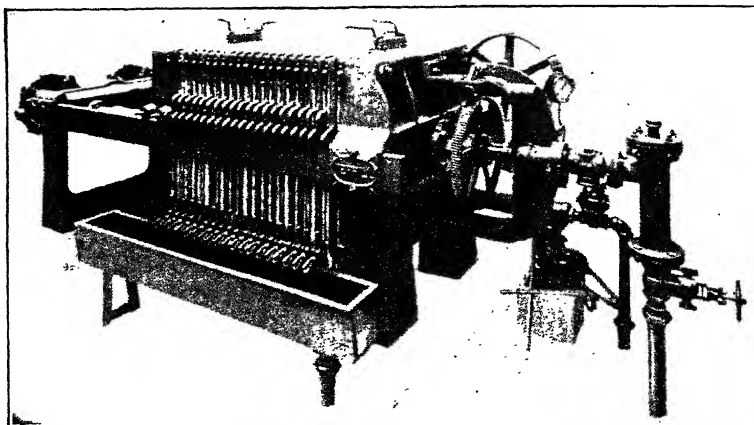


FIG. 16.—Merrill filter press.

Although at one time very popular in metallurgical work, the filter press is now being largely replaced by newer types of equipment. The apparatus, however, is simple in design and construction, and skilled labor is unnecessary. It has a distinct advantage in its adaptability to practically any filtration problem. Corrosive liquors can be readily handled, since the machine may be constructed of brass, aluminum, or wood.

The chief objection to the filter press is in the excessive amount of labor required for operation. Numerous instances could be cited where filter presses have been replaced by more modern types of machines with immense savings in operating cost. The life of filter cloth is less than for other types of machines, and this is a serious drawback because of the expense involved. Washing is not accomplished successfully in the filter press, since it is impossible to avoid short circuiting of the wash water. Filter-press operation nowadays exists only on a steadily diminishing scale.

Dehne Filter.—This type of filter press has found extensive application in many of the cyanide plants of South Africa and Western Australia. It is of the plate-and-frame type and differs from those just described in that the feed and wash water enter at the bottom through separate channels. Spigots at the bottom are provided for the

filtrate and conduits for the air and wash solution at the top. Washing is carried on by the through-washing process, the water entering behind the cloth of one plate, passing through the cloth and cake, and leaving through the cloth and discharge opening of the next cake.

Merrill Filter.—The Merrill filter is a development of the plate-and-frame press, whereby the cake is automatically sluiced out of the press, thus obviating the necessity for opening the press after each cycle of operations. The filter gained considerable recognition in the cyanide field and is still used in some plants, particularly the Homestake mill, where it was first developed.

Figure 17 shows the Merrill plate. The large conduit in the lower right-hand corner admits the feed and carries away the cake when it is sluiced away. Through

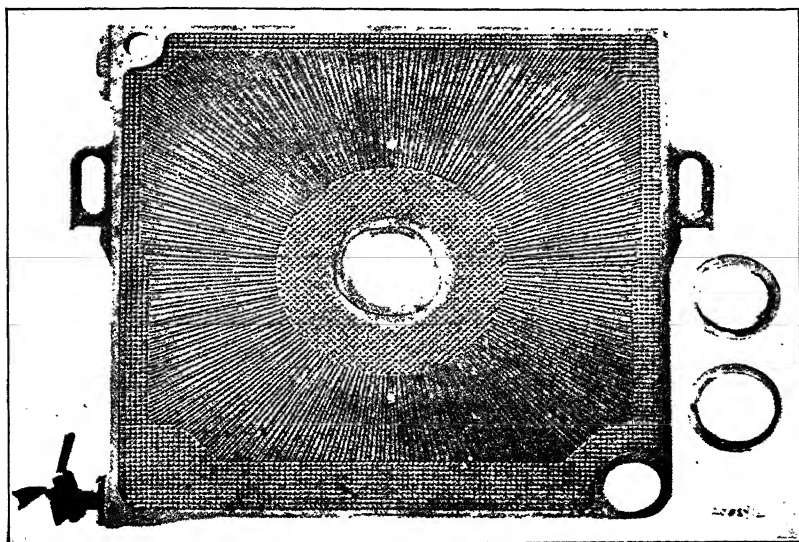


FIG. 17.—Plate of Merrill filter press.

the center of the press runs a rotatable sluicing pipe, operated by back gearing. The pipe carries a series of nozzles, one of which is provided for each compartment. Each nozzle is drilled with two holes, arranged so that the water will sweep the whole surface of the plate. The plates are corrugated with radial grooves, making the sluicing more effective.

In operation, the cakes on adjacent plates are not permitted to touch each other, leaving sufficient room between them for the nozzles to revolve. When the feed is shut off, wash water is introduced through the same conduit as the feed, washing by displacement. After washing, the press is drained, the sluicing pipe set in rotation, and water at a pressure of 50 to 75 lb. per sq. in. introduced through the pipe. This is permitted to continue until the press is thoroughly washed, usually requiring 5 to 10 min.

Inasmuch as the press is cleaned without opening, there is a decided saving in labor and filter cloth, and the capacity is greater, due to the saving of time in cleaning the press.

As the discharge is quite dilute, the Merrill filter is applicable only to those problems where the solids are waste products and dilution-water loss in the sluiced tailings is permissible. Expert operators are required to operate the machine, and the replacement of the cloth is troublesome since the press must be taken apart for this job.

Pressure filtration now finds application in a restricted field in metallurgical work. Large tonnages are handled by vacuum filtration, leaving a few specialized problems to various makes of pressure filters.

Suction-leaf Filter.—Moore's discovery and application of the principles of vacuum-leaf filtration was, at the time of its application, probably the greatest boon received by the cyanide industry since its inception. Prior to Moore's invention, filtration was carried out in large plate-and-frame presses that were especially troublesome to handle on slimy material and, in consequence, slowed up the whole process. Moore's main line of attack was to form a cake on the outside of a bag, so that the cake

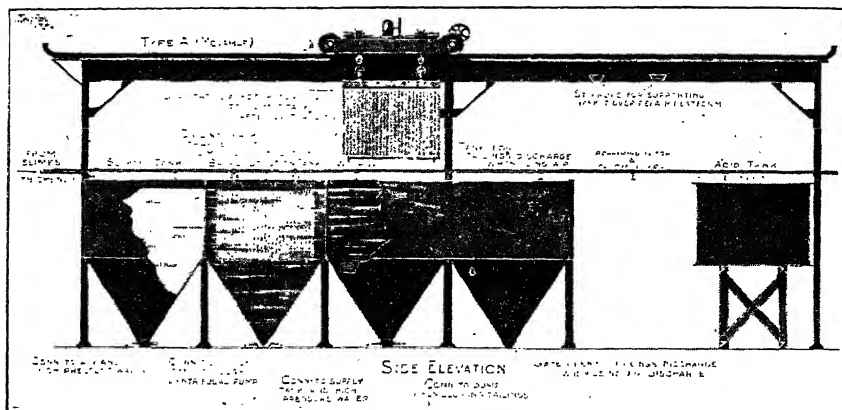


FIG. 18.—Moore filter.

could be dislodged by simply reversing the current through it. He also aimed to simplify the washing problem by keeping the cakes separate and thereby wash and discharge them independently.

The Moore filter (Fig. 18) consists of a number of rectangular filter leaves, clamped together to form a basket, and suspended in one of a series of four hopper-bottomed open tanks. The basket of filter leaves is moved and lowered successively into each of the tanks by means of an overhead crane. The first three tanks contain, respectively, the pulp to be filtered, weak solution, and water, and the last is designed to receive the discharge cake. Agitation of the pulp is obtained in the first tank either by blowing compressed air into the tank or by circulating the pulp with a centrifugal or air-lift pump.

Each filter leaf (Fig. 19) consists of a bag of canvas or other filtering medium covering a drainage member, and a framework to support the bag. The frame consists of a rectangular-shaped pipe, perforated at the bottom to supply the vacuum and to carry away the filtrate. The drainage member, whose function is to keep the bag from collapsing and to provide channels for the filtrate, is made either of corrugated screen fastened to the frame or of wooden slats slipped into pockets sewed into the

bag. The leaf is clamped at the top between wooden bars, which are attached to a pair of I beams, supporting the whole basket. The framework, forming the vacuum inlet to the leaf, is connected to a common manifold pipe, which is, in turn, communicative with a source of vacuum. A dry vacuum pump, receiver, and pump or barometric leg is the customary vacuum equipment.

In operation, the basket of leaves is first lowered into the pulp and the vacuum applied. The solids will, of course, be retained on the filter cloth, and the filtrate will pass through the cloth and out of the apparatus via the framework and manifold pipe. When the cake has reached the desired thickness, the basket is removed from the pulp tank by the overhead crane and swung to the next tank in series, containing barren solution. As the cake built up in the first tank is very uniform in consistency and thickness, an efficient displacement wash is obtained, by means of which the pregnant solution is forced out of the cake, leaving the barren solution. In order to recover the latter solution, the basket is removed from the second tank and transferred to the third, containing water. Here the process of washing is completed,

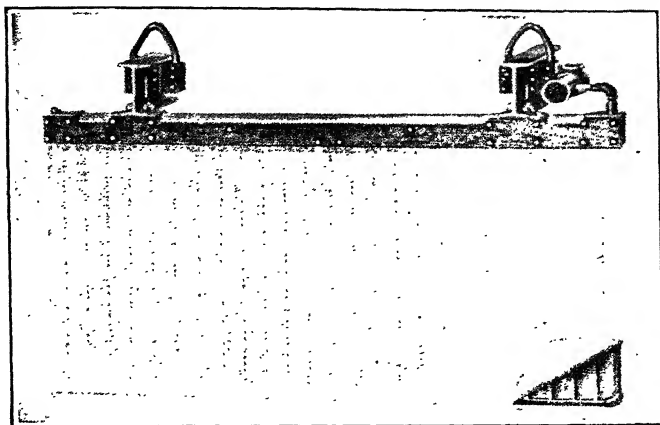


FIG. 19.—Moore filter leaf.

and the basket is raised out of the third and lowered into the fourth or discharge tank, containing no liquor. Here the suction is continued until the liquid content is reduced to the desired percentage. The vacuum is then cut off, and air pressure or steam is applied to the interior of the leaf by means of a suitable line. This pressure releases the filter cake, and it drops to the discharge hopper. The air current at the same time cleans the cloth preparatory to the next cycle of operations.

On an average cyanide slime, it requires 40 to 120 min. to form a $\frac{3}{4}$ -in. cake, requiring about 25 min. in the barren solution tank, about 15 min. in the water tank, and 5 to 20 min. to dry and discharge the cake. The time allotted in each tank depends, of course, on the rate of flow of the liquor through the cake at various intervals in the cycle, and this is usually determined by experiment or experience. The time required to transfer the basket from one tank to another is usually not over 1 or 2 min.

The filter leaves vary in size from 2×3 ft. to 8×10 ft. and are arranged in baskets containing 2 to 50 leaves, the space between the leaves being usually about 4 in. It is entirely practical to design a Moore filter plant with each unit or basket containing over 10,000 sq. ft. of filtering areas.

Another type of the machine has recently been introduced which employs a quantity of cylindrical cells instead of leaves for the filtering medium. These cells are all attached to a common header or vacuum chamber. The principles and operation of this type are identical with those of the leaf type.

The Butters filter is very similar in construction to the Moore, the sole difference lying in the use of one tank instead of four. The principle of the Butters filter consists in keeping the leaves stationary and changing the solutions in the same tank instead of transferring the leaves from one solution tank to another, as in the Moore. As a result, the initial cost of a Butters installation is appreciably less than a Moore, but the cost of operation is practically the same. Moreover, the necessity for extensive piping and numerous valves is apt to cause operating difficulties and occasionally the loss of valuable solutions.

The advantages of the suction-leaf filter over the plate-and-frame press are so numerous that its rapid acceptance by the metallurgical world is not to be wondered at. Perhaps the main advantage is the marked saving in labor. One or two operators can efficiently handle several units of immense filtering area where ten or more would be required for an equal capacity on presses. The ability of the Moore filter to give a high washing efficiency, due to the displacement wash, is an outstanding reason for its popularity. The life of the filter cloth is considerably longer than in a filter press, since there is no handling or wear between gasket surfaces.

Practically the only disadvantage of the Moore filter is that the difference in pressure is limited to that of the atmosphere, which is, of course, lower at high altitudes. As a pressure filter can operate at a pressure of 50 lb. or more, it can be readily seen that the filtering area need not be so large as that of a suction-leaf filter to perform the same task. With the perfection of the rotary vacuum filter as well as the pressure-leaf filter, the installations of the Moore type have fallen off markedly, although a few of the older plants have retained their suction-leaf machines.

Hamilton¹ gives the following data on suction-leaf filtration:

Name of mill	Average daily	Type of filter	Number of leaves	Size of leaf, feet	Total area, square feet	Area per ton per day, square feet	Time of cycle	
							Hours	Minutes
Nipissing (desulphurizing)	244	Butters	60	5 by 10	5,100	21.0		30
Nipissing (cyaniding)....	244	Butters	80	5 by 10	6,800	28.0		00
Waibi Grand Junction....	250	Moore	96	4.25 by 16	10,752	43.0		30
	(estimated)							
Tonopah Belmont.....	145	Butters	120	5 by 10	10,200	70.0		50
	(estimated)							

Pressure-leaf Filters.—The Kelly pressure filter (Fig. 20) consists of a horizontal press shell or pressure tank, supported on a heavy frame, and containing a traveling filter carriage supporting the filter leaves. While actual filtration is being performed, this carriage is telescoped into the tank and the tank is sealed with a closing device, forming the head of the chamber. To discharge the cake the head is opened, the carriage slid back, and the cake blown off.

The filter leaves are rectangular in shape and are arranged parallel to each other and to the longitudinal axis of the cylinder. They are all proportioned in size to conform to the cylindrical shape of the shell, and are connected individually to discharge cocks in the movable head.

¹ "Manual of Cyanidation," p. 232.

The locking device on the head works on the same principle as an umbrella. Radial arms are secured to the shell by V bolts, the locking action being actuated by a revolvable shaft and a toggle arrangement in such a manner that a small pressure applied to the shaft exerts a greatly magnified pressure on the radial arms. The mechanism is operated by a hand lever on the small machines and by an automatic hand or air-operated device on the large ones.

To provide for a constant positive pressure within the filter, an automatic air-regulating device is utilized, functioning on the motion of a float riding on the liquor in the filter. In order to effect a saving in floor space on the larger machines, the "twin-unit arrangement" is adopted, whereby two filters are placed with their movable

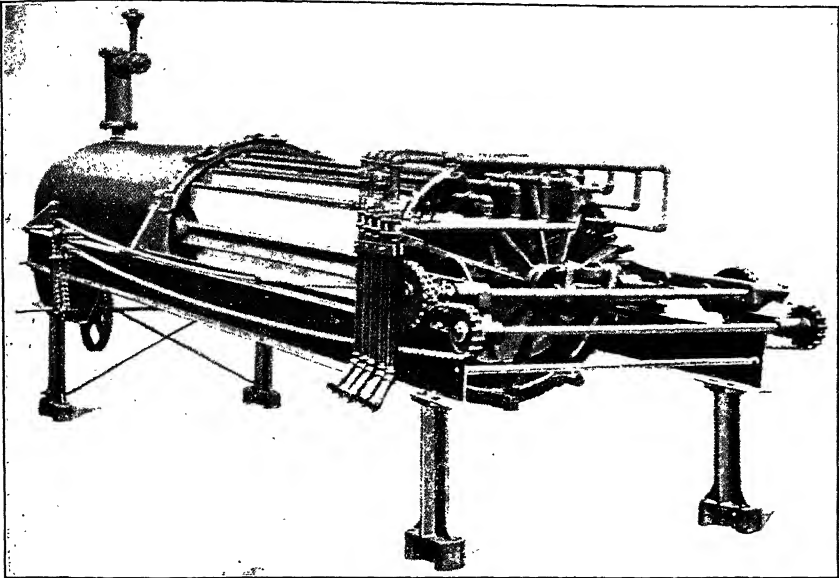


FIG. 20.—Hand-operated Kelly filter.

heads facing each other, each filter discharging over the same space, this operation, of course, being performed alternately by the two machines. The locking mechanism on both filters is operated by one air motor.

The operation of the Kelly filters is as follows: The carriage is drawn into the pressure chamber, locked, and the pulp forced in. The air escapes through a valve, and the clear filtrate flows from the cocks in the head to the discharge launder. As soon as cakes of sufficient thickness have been formed, the excess liquid is drained off, or forced out of the tank by compressed air. Wash water is then forced in until the cake is sufficiently washed, when the excess is drained off, and the cakes dried. To discharge, the head is unlocked and the carriage slid back, a slight slope to the framework assisting this. Compressed air, steam, or water is forced into the leaves to blow off the cakes. A set of counterpoise weights draw the carriage back into position, ready for the next cycle.

The Kelly is the simplest of all the pressure-leaf filters, although the operating cost is about the same. One man can readily operate two or more filters, depending on the length of the cycle. The capacity per unit surface is greater than of either the suction-leaf filter or plate-and-frame press, but is less than that of the continuous filter. Efficient displacement washing can be obtained by using a supply of turbid wash water, in order to fill in any gaps caused by the cake falling off.

Nowadays, the use of Kelly filters is found chiefly in chemical industry, and in petroleum refining, where its ability to withstand high pressures and to be insulated for large temperature ranges is especially appreciated.

Sweetland Filter.—The Sweetland, or clam-shell, filter (Fig. 21) is composed of two semicylindrical shells, the upper half being held rigid in a framework and carrying circular filter leaves with suitable outlet connections. The lower half is hinged to the

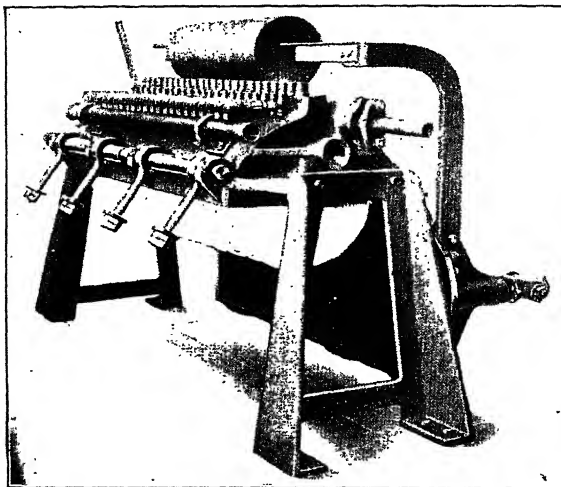


FIG. 21.—Sweetland filter.

upper, so that when closed and locked the two form a cylindrical pressure chamber, and when swung open the leaves are exposed.

The operation of the Sweetland filter is practically identical with the Kelly, the chief difference lying in the method of discharge. To effect this, the lower half of the filter is swung open and the cake blown off with a reverse current of compressed air through the leaves. As with all leaf filters, the discharge is not complete and some cake is retained on the leaves. The design of the filter is such that this remaining cake can be readily scraped off with a long paddle. In some cases, cake is now removed by sluicing, using a sluice pipe. This pipe is located inside the upper half of the shell, having nozzles that play on the surface of the filter leaves and thus wash the filter cakes away.

In the original design of the Sweetland filter, the pressure chamber was rectangular with a V-shaped bottom, and the filter leaves were square instead of round. Instead of opening the press and letting the cakes fall out, a sluicing arrangement was utilized. This design, however, was abandoned later and the present clam-shell design was adopted.

The Sweetland filter possesses all the advantages of the Kelly and suction-leaf types. In addition, the leaves are more readily handled than the Kelly's, and inspection of them is more conveniently accomplished. The ease of operation is unsurpassed by any other pressure filter. The fact that the filter is cyclic in operation is a drawback common to all machines of this type, and in consequence rotary vacuum filters have largely replaced leaf filters in metallurgical work, although they are still extensively used in the chemical industries.

Burt Filter.—Although the original design of the Burt filter employed filter leaves, the more popular type, the Burt revolving filter, possesses an individuality of design that really puts it in a separate class. It is of the self-discharge pressure type, but the filter medium is cylindrical. The Burt machine is designed distinctly for granular products and has found its greatest application in the *caliche* treatment plants of Chile, in El Oro district of Mexico, on cyanide pulp, and in handling corrosive pulps in electrolytic zinc plants.

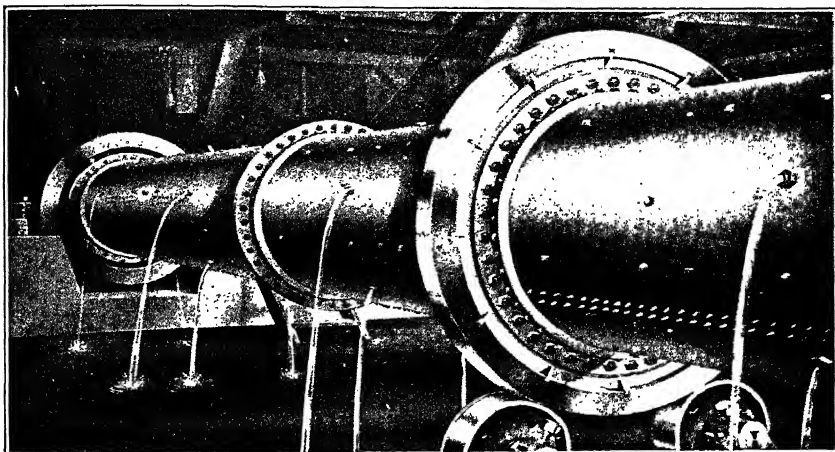


FIG. 22.—Burt filter.

The filter consists of a steel drum, similar to a rotary kiln, made up of sections, and rotating on a hollow trunnion, which also acts as the feed- and wash-water inlet. The interior of the drum contains a series of filter mats, secured to the inner shell by angles, so that solution from the pulp entering from the hollow trunnion passes through these panels and out through holes drilled in the shell. The end of the drum away from the inlet trunnion is closed by a cast-iron door operated by a hydraulic piston rod and a toggle joint and arms. The discharge of the cake is obtained by speeding up the rotation of the drum, which causes the cake to fall off, the solids being of necessity of a nonplastic nature. Spiral angle irons worm the cake along the drum to the discharge end of the filter.

The Burt filter is simple in construction and has an ease of operation not obtained in other pressure filters. The ability to form a uniform cake, irrespective of any changes in the settling rate of the solids, and the lack of necessity of any means of agitation are distinctly advantageous. Very efficient washing with a minimum of wash water is also obtained. A big disadvantage of the Burt filter is its inability to treat any but granular materials. The operator has a very troublesome job if the

material changes in character, so that only a thin cake can be built up. Also, the cake sometimes tends to hold together and will not fall off the cloth, and there is no provision in the design for taking care of this.

ROTARY VACUUM FILTERS

Oliver Continuous Filter.—The invention and development of the continuous filter produced a machine that showed as decisive improvements over other types as did the Moore filter over the old plate-and-frame presses. The increase in capacity, less labor required, and efficiency in washing were advantages of which mill operators were quick to take advantage. Although the Oliver was not the first of such machines, the great number in use entitles it to a leading position.

The fundamental idea of the rotary vacuum filter is to build up a thin cake and to stop this formation before the rate of flow of liquid through the cake has retarded appreciably from the maximum. This idea is in sharp contrast with the operation of the intermittent machines previously described, where it was necessary to build a thick cake in order to get a good capacity. As the cake on the Oliver filter is being

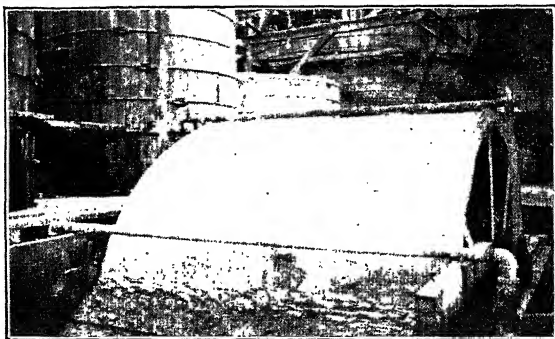


Fig. 23.—Original Oliver filter, North Star Mines.

continually formed, washed, dried, and discharged, with no lapse of time for draining and discharging the cake, it is obvious that the capacity will be greater than on a pressure filter, even though the difference in pressure is considerably less than can be applied to the other types.

The Oliver continuous filter consists essentially of a drum or cylinder rotating on a horizontal axis with the lower portion submerged in a tank containing the material to be filtered. The surface of the drum is divided into compartments or sections, the dividing partitions being parallel to the main shaft. These sections are covered with screen for supporting the filter medium, which is held in place and protected from wear by a spiral wire winding. This winding is usually placed with a space of 1 in. between adjacent wires. Each compartment of the drum is connected by means of pipes passing through a hollow trunnion to an automatic valve, which controls not only the application of the vacuum for forming and washing the cake, but also the admission of air for discharging. Thus, each compartment forms virtually an independent unit, although the filtering medium is attached as a continuous sheeting over the whole surface of the drum.

A scraper is fitted across the face of the drum and rests against the spiral wire winding in such a manner that the cake is entirely removed after being released by air or steam pressure.

An agitator operates in the tank beneath the drum for the purpose of keeping in suspension the heavier particles in the pulp and of ensuring a uniform cake. This agitator is of the oscillating type, provided with roller bearings mounted on the trunnions of the filter drum.

Washing of filter cake is accomplished by the use of drip pipes set near the surface of the cake, or by means of atomizing sprays suspended over the drum. Weak solutions and water washes may be independently applied.

The driving mechanism consists of a worm gear keyed to the trunnion on the valve end of the drum, meshing with a worm driven from a line shaft or individual motor, through a speed reducer. The speed of rotation of the drum depends upon the filtering characteristics of the material treated. It may be as fast as 1 r.p.m. or as slow as one revolution in 10 min. The power required to rotate the drum does not exceed 3 hp. and is 1 hp. or less on sizes smaller than the 11½-ft. diameter filters. Faster speeds or specially heavy duty may require somewhat higher horse power motors.

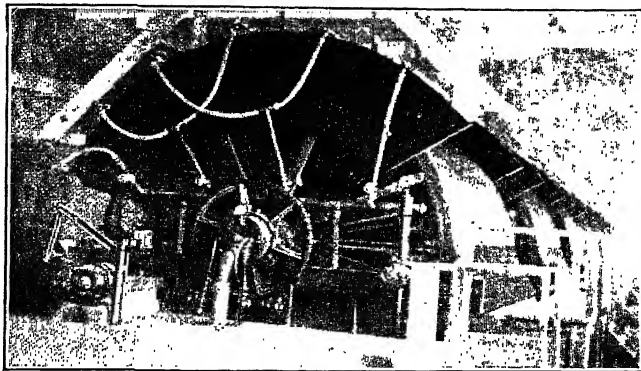
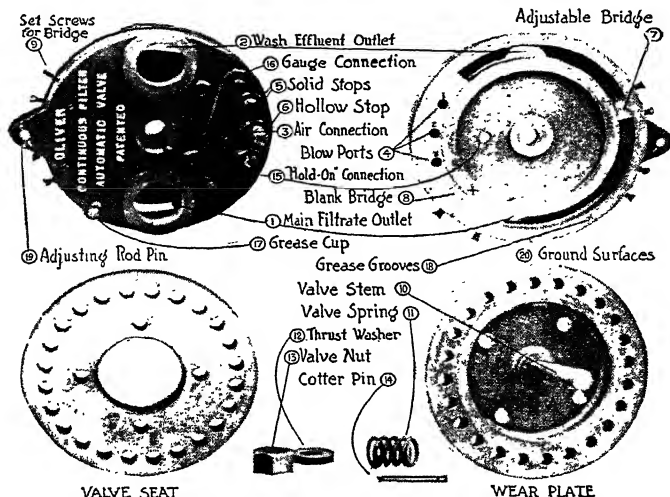


FIG. 24.—Late types of Oliver continuous filters at Hollinger Consolidated Gold Mines Co., Timmins, Ontario.

The Oliver valve is made with one or two solution outlets. The valve, shown in Fig. 25, is made with two outlets for the separation of the main filtrate from the wash water.

In the figure, the upper left-hand corner is an exterior and the upper right an interior view of the valve, the former side facing out from the drum. This part is stationary and is held in place by a valve spring (11) placed in the valve stem (10). Next to the valve comes the wear plate and inside of that the valve seat, both of these pieces being bolted to the trunnion and revolving with it. The discharge pipes from each section terminate in the holes in the wear plate. The large hole (1) at the bottom of the valve, as seen in the upper left-hand view, is the outlet for the filtrate. It will be seen that this opens to the annular port and is designed to carry off all filtrate obtained from the time when the compartment submerges until it emerges from the liquor. The upper hole (2) in the same cut is the outlet for the wash water and it, in turn, opens to the annular port. The two solutions are divided by the adjustable bridge (7). This bridge can be adjusted to any desired position or it can be removed altogether, the outlet (2) plugged, and all liquor taken off through the filtrate outlet (1). Compressed air ports (4) are provided, each controlled by a valve stop (5 or 6) which is adjustable from the outside of the valve. If the cake is easily

blown off, only the top port need be used, but if it is very sticky, or if the cloth becomes gummed up with oil or precipitates, the second or third ports can be used to prolong the blow. One stop (6) is hollow and threaded for pipe connection, so that steam may be used either at the same time as the compressed air or to follow the first blow. The stops are interchangeable so that steam may be admitted in advance of the air or can be used alone. In most cases today, a single blow port is provided and is sufficient for the purpose.



Courtesy Oliver Continuous Filter Company

FIG. 25.—Automatic valve of Oliver continuous filter. The upper right-hand cut shows the underside of the piece shown in the upper left corner. The valve is assembled by bolting the removable wear plate and valve seat to the end of the filter, each hole registering with a filtrate pipe from a compartment. The stationary member is centered by the valve stem and held against the wearing plate by the valve spring and nut. Rotation of this member is prevented by the adjusting rod pin.

The dry vacuum system is in most cases more efficient and satisfactory than the wet system. The vacuum receiver can be dispensed with in the case of the latter, but the former system is more flexible, durable, and efficient, besides being cheaper.

Over a period of years, various important and useful changes have been made in the design of the Oliver filter. For example, the familiar spiral wire winding on the filter cover has been discarded in the *panel-type Oliver filter*. In this variety of filter, the cover is held firmly at each dividing compartment, or division strip, on the filter surface by a suitable binder, which in some cases may be a metal rod, a fiber cord, or a rubber piece of circular section. These binders serve to force the cloth cover down into grooves lengthwise of the division strips, thus sealing each compartment, or section, from its neighbors. The edges of the cloth around the perimeter at the ends of the drum are held in place by a few strands of wire, or by Acme bands.

Operational benefits of this design are found in its ability to form and discharge easily thin cakes of slimy material. The filter cloth balloons out softly by a low-pressure air blow while the cake is removed readily by either of two new methods of cake removal adapted to Oliver filters.

One method is by the rubber-tipped scraper which carries a molded soft rubber strip with a tapered edge riding on the edge of the steel scraper. This rubber edge does not abrade the filter cover as the bare steel may, nor does it "groove" from the wire winding as steel scrapers will. The other method is by the taut wire stretched across the face of the filter. In some cases, the wire is barely tangent to the drum surface, and in others the wire is held close enough to lie on the cloth. This means of cake removal will discharge some thin cakes that are less suited to the use of scrapers.

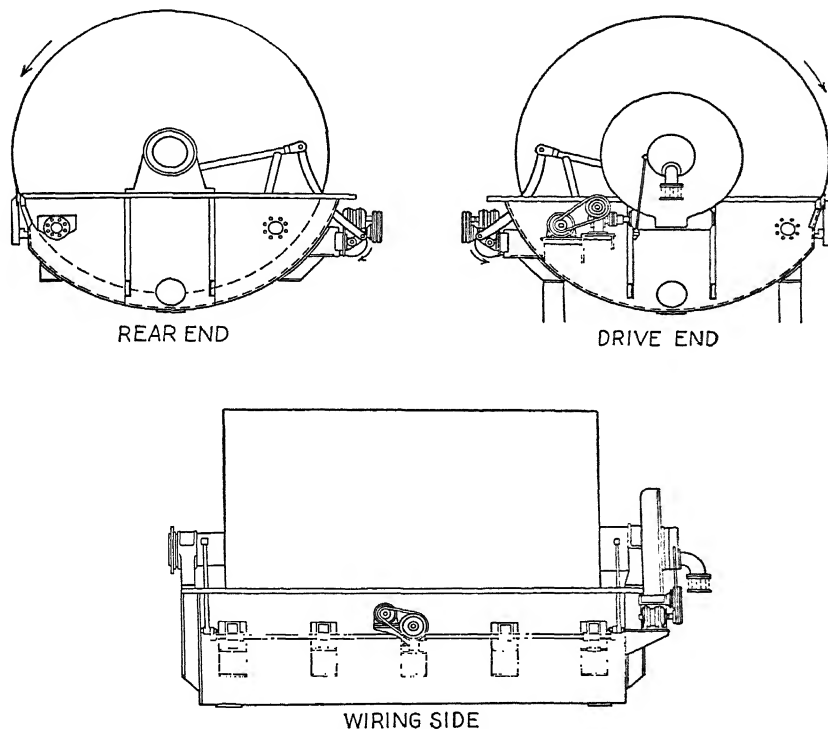


FIG. 26.—Oliver filter.

Recent use of the "low-submergence" Oliver filter has enabled a longer portion of the operating cycle to be devoted to aeration, washing, and drying of the filter cake. The conventional Oliver filter design provided 35 or 40 per cent of drum area immersed in the pulp, while the cake was discharged at a point 10 to 15 deg. above the horizontal center line of the drum.

In the low-submergence design, the cake is discharged at a point about a foot or more below the horizontal center line; in the largest sizes, discharge occurs 2 ft. below that point. Cake discharge is then aided by gravity pulling the cake away from the cloth surface. The low-submergence design is suited to any pulp that will form a cake readily in the shorter period of cake formation which that design allows. About 25

per cent of the filter area is immersed in the low-submergence design, when the depth of the pulp in the filter tank is at its greatest.

Shallow immersion and improved cake discharge of the thinner cakes, so formed, make possible maximum rotating speeds so that increased tonnages are often the result of using this design.

Other uses of rubber in modern Oliver filter design are found in sections of rubber hose in the internal drum piping; this relieves strains existing in an all-steel piping assembly.

A greater novelty and one of much promise is the minutely perforated rubber fabric, with 6400 holes per square inch, made by the United States Rubber Co. This fabric has been used with marked success on various metallurgical filter installations, giving increased tonnages, lower moisture in cake, and longer cover life than the usual cotton cloth.

Thus far, only the more granular, or moderately coarse, products are being handled by the rubber fabric. The economic preference for the rubber fabric requires it to last at least as long as cotton fabrics used over an equal cost interval. This would call, in most cases, for at least 6 months' operational experience.

The field of metallurgical filtration now covers important uses formerly unknown or of secondary importance, such as chromite recovery, electrolytic manganese plant work, recovery of alumina from various minerals and by different processes, and the recovery of magnesium compounds from minerals such as dolomite and brucite, as well as from sea water.

Improved washing of filter cakes is now possible on the Oliver continuous filter by use of the Oliver wash net. This accessory is a specially woven fabric that lies upon the surface of the cake on the rotating drum. The meshes of this fabric are filled by wash water dripped upon it so that wash-water contact is more abundant than before.

Synthetic fibers have now been converted to valuable service in filtration by the fabric known as Vinyon. This cloth is particularly useful in metallurgical filtration, as it is resistant to acid and alkali, but it is not suited to hot pulps.

Changing Filter Cloths.—The time required for changing drum-filter covers and the number of operators needed to make the change have both been reduced in recent years. For instance, the narrowly spaced ($1\frac{1}{2}$ to $\frac{3}{4}$ in.) winding wire, as formerly applied to the drum, may be compared with 1 to $1\frac{1}{2}$ in. as used nowadays; this cuts former winding time in half.

Better drive mechanism today permits faster drum rotation. The speed is now one rotation in 60 sec., which is faster than the winding speeds used in former years. For large drums, a winding device is now available for winding two wires simultaneously onto the drum. All these factors combine to make a large reduction in time needed for placing a new cover on the filter. A large filter can be recovered and returned to operation in a half-shift. For this work, it is best to have one man tending the wire that is being wound onto the drum face, while a helper is usually watching the unwinding of wire from the reel.

Removing the old cloth and applying the new cloth on drum, before winding the wire, requires a couple of hours. For this, two men are needed for a small drum and three or four for the longer drums, depending on the length. Skill and experience count here as in every task requiring manual dexterity.

The modern panel-type Oliver filter does not use wire winding, the filter cloth being calked into grooves in the longitudinal division strips, while at the ends of the drum, the cloth is clamped by special bands. Naturally, this design provides a much quicker change of cloth. In view of longer cloth life nowadays, often 150 days and upward, the percentage of operating time lost by change of cloths is a small fraction of 1 per cent of the total running time.

Worldwide use of the Oliver filter in the filtering and washing of cyanide pulp has made it a standard element in cyanidation.

Its objective and performance therein have undergone gradual changes so that it is now widely used as a means for rapidly extracting undissolved gold, as well as a means for reducing mechanical solution losses in the tailings.

Many cyanide plants have provided double (or "series") filtration of the pulp underflow from the preceding thickening, which usually means, nowadays, two thickenings, followed by two filtrations, as the best modern practice for most gold ores.

Processes in which the cake solids are the valuable product rely on the thickener-and-filter team to deliver a final cake with the least moisture economically obtainable.

All the base-metal sulphides recovered by flotation are so handled today, as well as many other nonsulphide minerals, among which are washed fine coal, various nonmetallics, electrolytic-process products, and a varied group responsive to filtering.

The following table gives the standard sizes of Oliver metallurgical type filters with the respective filtering areas:

Diameter	Length	Area, sq. ft.	Diameter	Length	Area, sq. ft.
3' 0"	0' 6"	4	8' 0"	10' 0"	250
3' 0"	1' 0"	9	8' 0"	12' 0"	300
3' 0"	2' 0"	18	11' 6"	12' 0"	430
3' 0"	4' 0"	36	11' 6"	14' 0"	500
5' 4"	4' 0"	65	11' 6"	16' 0"	570
5' 4"	6' 0"	100	14' 0"	14' 0"	610
5' 4"	8' 0"	135	14' 0"	16' 0"	700
5' 4"	10' 0"	165	14' 0"	18' 0"	790
8' 0"	8' 0"	200			

Capacities.—The capacity of an Oliver filter is largely dependent on conditions, and it is impossible to give figures that will hold for all cases. The physical condition of the solids, whether granular or colloidal, the dilution of the feed, the size of the particles, the pH value, and the temperature of the pulp are all influencing factors. When handling a very colloidal and slow-filtering cyanide slime, the feed containing 40 per cent of solids, the filtering rate may be in the neighborhood of 200 lb. per sq. ft. per 24 hr. On the other hand, from a granular slime containing 50 per cent of solids, a rate of 600 to 2000 lb. and, in extreme cases, as much as 2500 lb. is obtainable. On flotation concentrates, the range is 200 to 2000 lb., with a probable average of 600 lb. per sq. ft. per 24 hr. In each case under consideration, a filtration test should be made, the facilities and experience of the manufacturer being available for this purpose.

The pronounced advantage of the Oliver filter is the saving in labor, a whole battery of filters requiring less than one man's time per shift. Replacement of cloth, of course, takes time and labor, but, in general, the cloth is replaced easily and quickly. Washing the cake is more uniform than in any of the intermittent filters owing to the eque-resistant condition of the cake and the definite time allotted to washing. The high capacity obtainable because of the continuous cycle of operation is a marked advantage over pressure filters. The convenience in observing the work being done is not obtainable with the former machines as it is with the Oliver, and this factor, of course, promotes efficiency of operation.

Performance data on filtering and washing shows that the moisture content of the discharged cake varies, as the objectives (washing of cake vs. the delivery of minimum

moisture cake) are unlike in conditions. Likewise, screen analyses also differ, and the mineral composition of the pulp, whether oxidized, sulphide, siliceous, or clayey, confers quite varied qualities to the respective cakes. All these metallurgical pulps are handled by continuous filtration today, by a judicious control of working conditions. The choice of filter type is directly influenced through these inherent differences in the pulp to be filtered.

American Filter.—The American filter (Fig. 27) differs from the Oliver chiefly in the design of the filtering surface. Instead of a drum, this consists of a series of vertical disks fixed side by side on a rotating central shaft with the lower parts of the disks submerged in the pulp to be filtered. Although differing in design, the general principles of operation of the drum filter are retained.

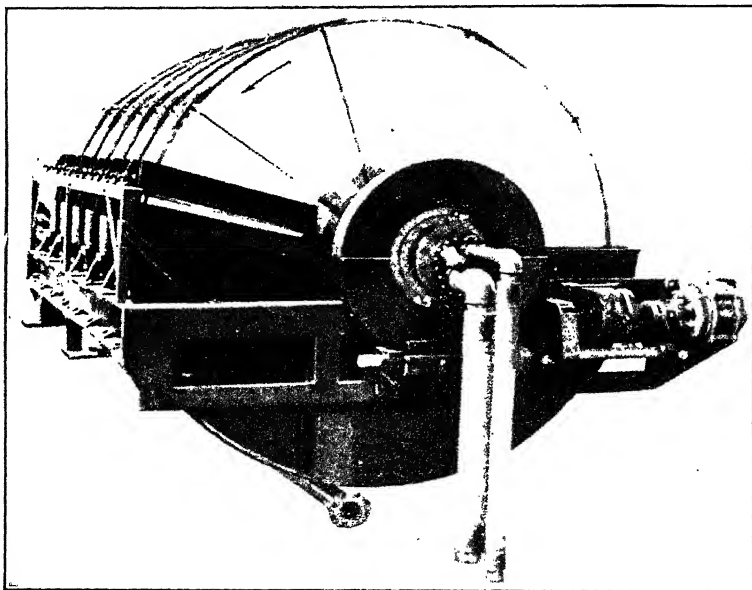


FIG. 27.—American filter.

As shown in Fig. 29, each disk is made up of a number of sectors, this number depending on the size of the machine. Each sector is made up of corrugated wood backing, suitably reinforced and inserted in a drainage cap, and is covered with a filter bag, fastened with a clamp. The sector is then fixed in place on the central shaft.

When assembled on the shaft, the corresponding sectors of all the disks have a common filtrate channel, there being as many of these channels as there are sectors in the disks. These channels cored into the cast-iron segments of the central shaft all lead to the rotating seat of a filter valve, which is the same as the Oliver valve.

Inasmuch as the cake is formed on vertical surfaces, a specially designed filter tank is required. As shown in Fig. 30, the tank is made in two parts, the right-hand side being the cake-forming compartment, and the left-hand side forming the indi-

vidual compartments for the cake discharge. The cake is removed by a scraper and falls between the pans to a longitudinal hopper.

The operation and the layout of the American filter are so nearly identical with the Oliver that a detailed description is unnecessary. Replacement of the cloth differs from that of the drum filters. The sectors are all replaceable, and repairing a defective leaf simply means removing the old sector and replacing with a new one. Meanwhile the cloth on the removed sector is taken off and new cloth is put on.

The chief advantage of the American filter, and the one that was the basis for its design, is the increase in area over drum machines for a given unit of floor space. One

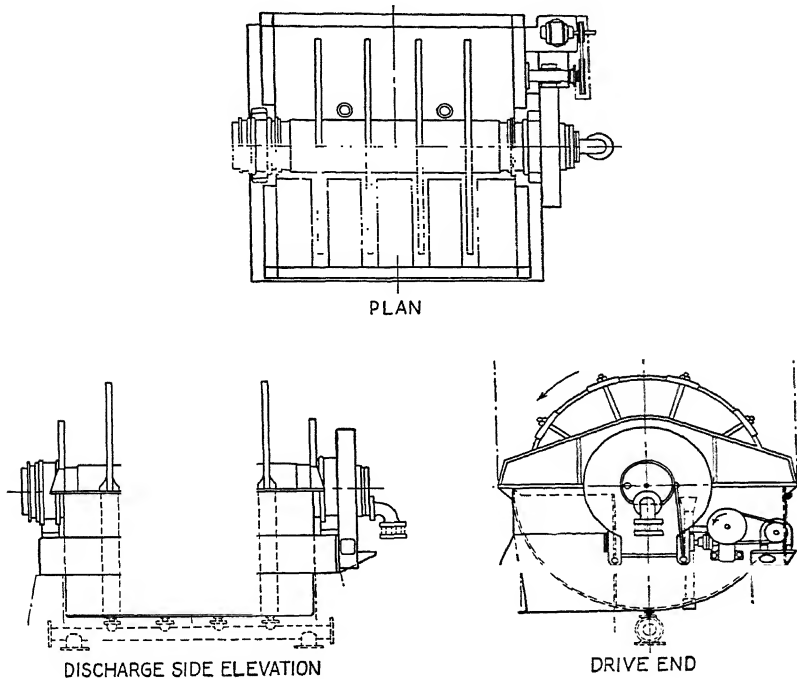


FIG. 28.—American filter.

machine can be made with an area greatly exceeding that obtainable with the drum filters. Replacement of sectors of the disk in order to repair the cloth is very readily accomplished, and the lack of necessity for wire winding is a timesaver.

The cake formed on the American filter sectors is removed by reverse air pressure inside the filter bags covering each sector. This pressure is low, usually provided by a 1-lb. air-pressure blower, or the like, and as the inflated bags rotate past the discharge point, the cake is removed either by a curved-blade cake discharger or by a fluted radial roller (in case of some sticky cakes) which lies against the cake surface.

The rollers are positively driven from a toothed gearing on the outer rim of the disk, so that roller and filter disk both rotate at same speed. Flutings on the roller fill with cake, and this cake in the flutings adheres to the cake on the disk in sub-

sequent rotations. The cake adhering to the roller is sliced off by a knife fixed in a tangential position to the roller and then drops into the cake-discharge chute.

The standard diameters of the American metallurgical filters are 4, 6, $8\frac{1}{2}$, and $12\frac{1}{2}$ ft. Large areas are obtainable in the latter two diameters, up to 1125 sq. ft. in the former and 2400 sq. ft. in the latter. Present practice assigns the American filter

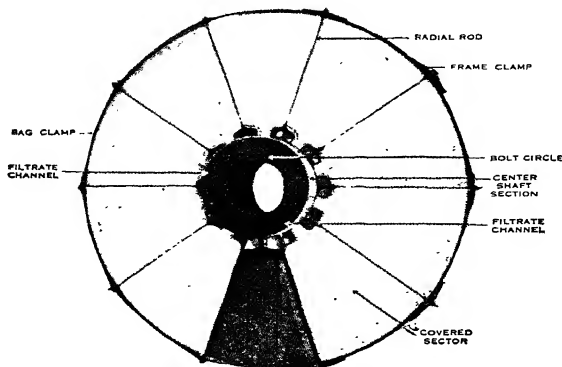


FIG. 29.—Leaf—American filter.

chiefly to dewatering duties rather than to those requiring washing of the filter cake. Like the standard Oliver design, the American leaves the coarsely granular products to be handled by the Dorco filter, the hopper dewaterer, or the Oliver sand table.

The FEinc Filter.—This design follows the general design of the Oliver standard design except for the method of cake discharge. A large number of closely spaced small endless cords or string pass around the filter drum upon the cloth and lead away

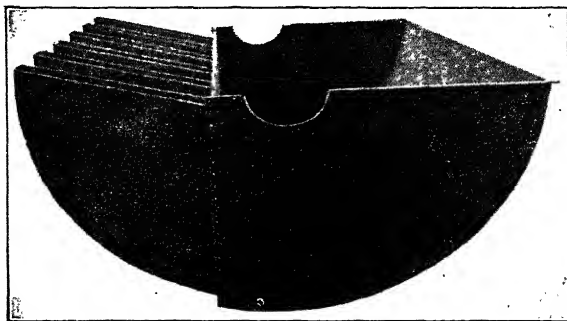


FIG. 30.—Filter tank—American filter.

from it around a snubbing roll, parallel to the axis of the filter drum, before returning again to the drum surface.

As the cake is formed on the cloth and cords, the latter carry the cake with them as they pass around the snubbing roll. The cake then breaks away from the cords and falls away from the filter. This construction has led to the use of this filter design for handling very thin and sticky cakes, such as those in china clay production.

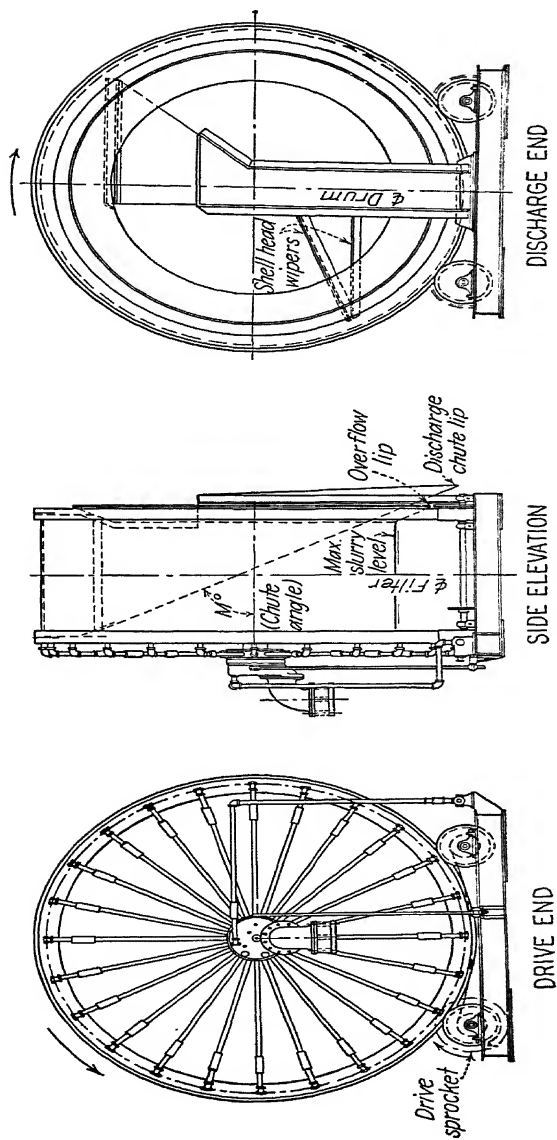


FIG. 31.—Dorco filter.

To date, this design has not had wide use in the metallurgical field, though some cyanide plants with very clayey pulp have used FEinc filters.

For the granular products unsuited to the previously described designs, other types are available. The production of these sandy granular pulps has much increased, and as their value justifies economical continuous dewatering, they are now being handled by three types of filters. These are the Dorcco filter, the hopper dewaterer, and the Oliver sand table, or horizontal rotary vacuum filter. Invented at different times by different people, all three have been improved in design lately and are now handling much increased tonnages of mineral products.

The Dorcco filter is a continuous drum filter, whose filtering area covers the internal surface of the rotating drum instead of the external surface as in the case of the Oliver filter. It has no tank in which this drum rotates as the pulp feed is held in the drum itself by the low curb ring that borders the open end of the drum.

Vacuum is applied to the undersurface of the filtering area by an automatic control valve and piping as in the case of the Oliver filter. This vacuum causes formation of a cake upon the cloth surface as it rotates past the pulp which fills the lowest part of the drum.

As rotation continues, the cake, on the filter surface, after emerging from the pool of pulp is further dried: when it reaches the zenith of the drum, vacuum is shut off by the automatic control valve, and the cake drops off. Back-pressure air is often not needed for dropping the cake at this point.

To discharge the more adherent cakes, alternate blowing and suction at the discharge point may be applied by an air-blow valve.

The Dorcco filter has been widely adopted for handling heavy concentrate such as magnetic iron concentrates in the New York and New Jersey fields, and other similar mineral products, relatively coarse in mesh and free from slimy nonmineral constituents.

Capacities of 2 or 3 tons per sq. ft. of filter area daily are obtainable, and moisture in the discharged cake is low, depending on the feed material. However, 4 or 5 tons may be attained in good conditions.

Some heavy slime-free moderately granular mineral products require both filtering and washing, and for this problem the Oliver sand table is available. This machine is a flat, annular filter surface, whose function is controlled by an automatic vacuum valve like the Oliver filter, with piping that extends from the valve to the several sections of the filtering surface.

The annulus is provided with suitable rims which retain the filter pulp, and after the application of wash liquid, the washed and dewatered cake is removed by different methods. They include a plow, a revolving scroll, or a drag-flight conveyer.

Some very interesting and recent applications of this design are the dewatering and washing of a gypsum and phosphoric acid mixture, requiring thorough washing and production of undiluted "first filtrate." Another is the removal of surface liquid from a silica gel and acid mixture, without cake washing.

These plants may not be identified otherwise at this time, as they are related to the war effort. As is evident, the sand-table units noted above are built to resist the corrosive solutions they handle, using lead, plastics, and alloy steels as required. A wider extension of use for this serviceable unit is expected in dewatering fields as well as in washing problems.

Rotary-hopper Dewaterer.—The function of this machine is to dewater granular products such as crystals. It consists of a number of hoppers mounted on a horizontal shaft. Each hopper has a filter-cloth bottom, suitably supported, making each compartment a separate unit.

The feed is dropped into the hoppers on the ascending side at a point about 70 deg. below the zenith of travel, and vacuum is applied at the same time. Suction continues until the hopper reaches the horizontal position, when air or steam may be introduced to discharge the cake. Usually this is not needed, as the cake falls when vacuum is cut off.

The rotary-hopper dewaterer is a simple machine for dewatering granular products, and as it is continuous and automatic, it has an advantage over gravity bed filters. Its application, however, is limited to only a few products. The fact that the machine is idle for two-thirds of a revolution limits its capacity and efficiency.

Interesting applications of this machine are the dewatering of a willemite product at Franklin, N.J., and chromite concentrate at Benbow, Mont.

Operating Data.—The following tabulation gives data on filter operation at various typical mills. It has been found to be exceptionally difficult to obtain up-to-date authoritative data, and at the same time have it complete. Consequently,

Property	Idaho Maryland Gold Mine, Grass Valley, Calif.	Madsen Red Lake Gold Mines, Ltd., Red Lake, Ontario	Wright Hargreaves Gold Mines, Ltd., Kirkland Lake, Ontario	Hollinger Cons. Gold Mines, Ltd., Timmins, Ontario
Pulp solids.....	Flotation conc. (A) Cyanide pulp (B) (sulphides)	Reground siliceous cyanide pulp	Reground siliceous cyanide pulp	Reground siliceous cyanide pulp
Feed density, % solids	(A) 62% (B) 50%	57%	50%	52 to 55%
Per cent through 200 mesh.	(A) 92% (B) 100%	77%	40% is -10 microns	75%
Filter type.....	Oliver	Oliver	Oliver	Oliver
Size.....	(A) 5' 4" × 6' (B) 8' × 6'	11½ × 12' (A) Cyaniding 11½ × 16' (B)	(A) four 14 × 16' 1st stage (B) four 14 × 16' 2d stage (C) one 11½ × 14' Flot. conc.	(A) six 14 × 16' 1st stage (B) nine 14 × 16' 2d stage
Number in use.....	(A) one (B) one	(A) one (B) two		
Tons per sq. ft. filter area per 24 hr.....	(A) 0.84 (B) 0.32	(A) 0.92 (B) 0.70	(A) 0.57 (B) 0.57	(A) 0.81 average (B) 0.54 average
Type of cloth used.....	Cotton	Seythes Co. No. ST-10 cotton	20-oz. cotton twill	Cotton twill
Life of cloth.....	12 months	195-200 days	60-90 days	200-275 days
Cake moisture.....	(A) 21% (B) 22%	(A) 21% (B) 23%	22%	22%
Cake thickness.....	¾"	(A) ½" (B) ⅝"	½-⅝"	⅞-1"
Filter speed.....	(A) 3 min. per rev. (B) 6 min. per rev.	(A) 3.66 min. per rev. (B) 3.0 min. per rev.	3.25 min per rev.	3.03 min. per rev.
Vacuum used.....	24"	23"	26"	22-26"
Pulp temp.....	Room	70°F.	80°F.	80°F.

Data established in 1939-1940.

many of the spaces in the tabulation are left blank, because the specific information desired was found lacking.

Costs.—A compilation of average costs for machinery has value as a guide in preparing estimates for installations.

These average costs are assumed to be based chiefly on a stable average price level for construction material and factory labor.

Since the level of these factors is certainly not now considered to be at a well-stabilized point, any reliance on wartime costs is not justified as a basis for predicting future equipment costs.

Beattie Gold Mines, Ltd., Duparquet, Quebec	Sherritt-Gordon Mining Co., Sherridon, Manitoba	Falconbridge Nickel Corp., Falconbridge, Ontario	Andes Copper Mining Co., Potrerillos, Chile	Tennessee Copper Co., Copper Hill, Tenn.
Calcined flotation concentrates (A) Dewatering (B) Cyaniding 1st and 2d stages	Flotation concentrate chalcoppyrite	Flotation concentrate	Flotation concentrate	Flotation concentrate Cu, Fe, Zn
(A) 50% (B) 48%	50%	55-65%	72-79%	Cu 75% Fe 80% Zn 40%
(A) 61.0% (B) 99.0%	61.2%	58% (considerable slime)	60.6%	Cu 83% Fe 78% Zn 88%
(A) 11½ × 14' Oliver (B) Two 11½ × 14' Olivers	6' 6" Disk American filter	Two 6' 6" disk American filters	Four 14' × 13' Olivers	Cu 6' 2" Disk Fe 6' 3" Disk Zn 6' 1" Disk All Americans
0.345	0.6	0.32	0.367	Cu 0.84 Fe 2.6 Zn 0.24
32-oz. cotton twill	Cotton twill	Cotton twill	Filter fabrics No. 26	Cotton twill
(A) 90 days (B) 60 days	30 days	60 days	83 days	Cu 30 days Fe 15 days Zn 30 days
		11-13%	9.5-12.0%	Cu 10% Fe 9% Zn 8%
		½"	¾-1"	Cu ¾" Fe 1¼" Zn ½"
		5 min. per rev.	5.83 min. per rev.	Cu 10.0 min. per rev. Fe 2.7 min. per rev. Zn 10.0 min. per rev.
		25"	16"	25"
		85°F.	53°F.	70°F.

Data established in 1939-1940.

Therefore the convenient comparison of costs found in the first edition of this book cannot be compiled accurately in today's economic shifts amid wartime changes.

Such a study will find a sounder basis in postwar conditions.

Cost of Filter Operation.—The important economic factors in operation may be classed as follows: (1) investment charge, (2) depreciation charge, (3) cost of power, (4) cost of direct labor, (5) cost of maintenance and supplies, and (6) comparative percentage recovery of metal by alternative equipment.

The customary practice for computing the investment charge is to figure 6 or 7 per cent annually, based on the idea that the money would earn that rate if invested outside the business. However, the money should earn much more than that, at

San Francisco Mines of Mexico, Ltd., S. Francisco del Oro, Chihuahua, Mexico	St. Joseph Lead Co., Edwards, N.Y.	U.S. Smelting, Refining & Mining Co., Midvale, Utah	Mufulira Copper Mining Co. Ltd., Northern Rhodesia, South Africa	Cons. Mining & Smelting Co., Trail, B.C.
Flotation concentrate Lead, zinc	Flotation concentrate Pb, Zn, Fe	Flotation concentrate Fe, Zn, Pb	Flotation concentrate Cu	Flotation concentrate Pb, Zn
Pb 78% Zn 71%	Zn 43%	Fe 65% Zn 60% Pb 70%	56%	Pb 78.9% Zn 49.8%
Pb 69.7% Zn 59.0%	Pb 91.4% Zn 54.4% Fe 41.0%	Fe 22% Zn 18.6% Pb 10.2%	87%	Pb 69.9% Zn 55.4%
Pb one 4' 6" disk American Zn one 4' 6" disk American	Pb 8 × 14' Dorrego Zn 8' × 14' Dorrego Fe 12' × 16' Dorrego	Fe two 6' 19" disk Amer. Zn two 6' 10' disk Amer. Pb one 6' 10' disk Amer.	Two 10' × 4' Dorregos	One 8 × 12' Oliver Six 6' 6" disk American
Pb 1.35 Zn 1.49		Fe 0.336 Zn 0.288 Pb 0.720	0.26	Pb 1.44 Zn 1.00
No. 175-TW	No. 26	Cotton twill		
26-29 days	42-138 days	Fe 25 days Zn 30 days Pb 60 days	20-40 days	60-103 days
Pb 7.5% Zn 7.7%	Pb 7.6% Zn 7.4% Fe 9.5%	Fe 11.0% Zn 9.5% Pb 10.0%	9.8%	Pb 8.5% Zn 9.5%
34"	56-34"	Fe 3½" Zn 3½" Pb ½"	13-1"	Pb 34-1" Zn 1½"
3.5-7.0 min. per rev.	4.5-6.0 min. per rev.	8-1 min. per rev.	5.0 min. per rev.	Pb 4.83 min. per rev. Zn 0.58 min. per rev.
19"	16-18"	24"	22"	23.5"
Room	42-78°	Room	78°	66°

Data established in 1939-1940.

least 20 per cent, to compensate for the business risk involved. Although only 6 per cent should be charged, the earnings of 20 per cent or more should be kept in mind in the purchase of equipment, and the additional investment in the choice of a more expensive machine must be justified by a definite increase in the net profit of operation. On this question, the plant manager must establish his own set of assumptions.

In any discussion of costs, from the plant manager's viewpoint, the economic results from the use of the apparatus must enter into the cost formula.

For example, if the loss in recovery when using one type of machine exceeds that incurred when using another type, whose installed cost is more than that of the first type, then a true cost comparison must include both available recovery as well as cost.

Improvements in design and materials make such inherent betterments in results that comparisons of costs are made even more difficult.

For example, new kinds of filter fabrics form cakes more easily, and discharge them more completely and ensure lower maintenance than the conventional cotton fabrics that formerly were used almost exclusively.

First Cost of Filters.—The wide variety of types and sizes, as well as materials of construction, causes a like variation in prices so that a common price per unit of filter area is impossible.

For simple dewatering types, built of wood, steel, and cast iron, the price per square foot of filter area would be about \$16 to \$18 in the smaller sizes and \$10 to \$12 in the larger sizes.

With special designs in wood for acid resistance and those built for handling caustic pulps, the smaller sizes may cost about \$30 to \$40 per square foot of filter area and the larger ones about \$20.

In some special cases that embody various costly alloys, rubber-covered surfaces, or special metals of construction, there is a much increased cost.

These tentative prices were assembled under the conditions existing before the outbreak of the Second World War. Wartime conditions produce misleading indices of cost.

Operating Costs.—These vary widely with the tonnages handled and the nature of the feed sent to the filters.

Very low costs, \$0.020 per ton, have been consistently obtained over long periods, where conditions were favorable and tonnages were large.

Installations handling moderate tonnages and without benefit of specially favorable conditions have shown costs of \$0.05 to \$0.15 per ton of filter feed, varying with local conditions.

Favorable conditions include little or no colloidal material in the feed and a relatively dense feed to filter. Warm pulp is a decided aid to increased filtration rate, and so is a well-flocculated pulp.

From these factors giving higher output, it is also possible to obtain a lower filtering cost.

Calculation of operating costs should contain only those items directly pertaining to actual work. Fixed charges should not be prorated but should be carried as a separate item because the inclusion of fixed charges in operating costs will distort the true figure and make cost figures misleading in comparison with other similar operations.

Sometimes filtering is combined with thickening and agitation to give a so-called "treatment" charge in cyanide plants. The resultant figure may be interesting to the local management but has little value to other operators who may seek a comparison of unit cost figures.

Filter Presses.

Depreciation: 5 to 10 per cent per year, dependent on nature of liquor.

Power: 0.005 hp. per sq. ft. of filter area.

Labor: two men required to handle two presses with 2-hr. cycle.

Maintenance, exclusive of cloth: 3 to 5 per cent of investment.

Cloth: life is 3 to 4 weeks at 10-hr. day operation.

Pressure-leaf Filters.

Depreciation: 5 to 10 per cent per year.

Power: 0.005 to 0.02 hp., dependent upon head and capacity.

Labor: two men to handle four or five machines, each having 1000-sq. ft. area.

A foreman and mechanic will also be required for a large installation.

Maintenance, exclusive of cloth: 3 to 5 per cent of investment.

Cloth: life is 6 weeks to 6 months at 10-hr. day operation.

Continuous Vacuum Filters.

Depreciation: 5 to 10 per cent per year.

Power: 0.05 hp. per sq. ft. of filter area.

Labor: One man for six machines, or more.

Maintenance, exclusive of cloth: 5 per cent of investment.

Cloth: will last 4 to 12 months:—cotton cloth covers.

Wool cloth, 1 week to 6 months.

Metal cloth, synthetic fiber also used, as well as rubber and glass fiber.

Centrifugal Separation.—Innumerable attempts have been made in the past few years to apply the centrifuge to metallurgical operations, but the chief difficulty has been in making these machines continuous in action. A marked increase in efficiency of operation must be shown to justify the increased cost of operation. The Carpenter centrifugal drier has achieved some success in dewatering $\frac{3}{8}$ in. \times 0 washed coal, and the Bird centrifuge has done good work in effecting unusually close separation of fine granular material from slime. This is, more correctly speaking, a sizing or classification operation rather than a filtering operation.

The Bird centrifuge is an outgrowth and later development of the machine first known as the Laughlin. Removal of the solids thrown to the outer wall of the machine by centrifugal force is effected by a rotating spiral screw that rotates at a somewhat slower differential speed.

The fundamental formula for centrifugal force is

$$F = \frac{Wv^2}{gv} = 0.000341WRN^2$$

where W = weight in pounds.

R = radius in feet.

N = revolutions per minute.

The derivation of this formula can be obtained from any textbook on mechanics.

It is obvious that the best way of increasing the centrifugal force is to increase the speed, using as large a bowl as possible. The strength of a large machine, running at high speeds, however, is less than its bursting pressure and therefore the ordinary centrifuge of the Sharples type employs a small basket about 4 in. in diameter and 36 in. long, running at 20,000 to 40,000 r.p.m. This machine, however, has a small capacity and would hardly be applicable to the immense tonnages of the metallurgical industries.

CHAPTER VIII

METALLURGICAL FUELS

BY DONALD M. LIDDELL¹

Metallurgical Fuels.—The metallurgical fuels are coal, coke, wood, charcoal, peat, oil, and gas, the latter being both natural and manufactured. Peat is not important metallurgically. The other fuels will be discussed in the above order.

Coal is usually classed as anthracite, semianthracite, semibituminous, bituminous, subbituminous, and lignite. Roughly, these classes follow an increase in the ratio of volatile combustible to fixed carbon, but any strictly chemical classification will prove unsatisfactory, for the physical characteristics are fully as important as the chemical analysis, so that both must be considered in classifying a coal. In fact, the classes merge gradually into each other so that hard-and-fast lines cannot be drawn.

Anthracite is a coal having a ratio of fixed carbon to volatile combustible of over 10:1. At one time it was used to some extent in blast furnaces, instead of coke, but it does not loosen the charge as coke does. Because of the higher prices paid for large-sized anthracite for domestic use, and because of its short flame, anthracite is a very unimportant metallurgical fuel today. Some metallurgical power plants, however, burn the small-size anthracite under forced draft for steam raising. Semianthracite coal usually has a ratio of fixed carbon to volatile combustible of 6:1 to 10:1. It also is negligible metallurgically.

Semibituminous coal has a similar ratio of 3:1 to 7:1. Of all coals it has the highest calorific value for a given weight, and is little used in nonferrous metallurgy. It is practically smokeless and is much sought after for naval use. It is very friable, and if it has to be transported for any distance it shatters badly. It is best burned with mechanical stokers. Bituminous coal has a ratio of fixed carbon to volatile of under 3:1, as has also subbituminous coal and lignite, but bituminous coal weathers only slightly and into prisms, while subbituminous and lignite weather quickly and into thin plates.

Bituminous coal as a rule cokes well, but some of the semibituminous coals make a good coke, and some of the subbituminous coals also will produce coke, so a coal must be tested in order to determine whether it will coke or not.

Cannel coal is a variety of bituminous coal composed almost entirely of spores, spore cases, seed coats, etc., of the giant ferns of the Carboniferous age. Woody material is almost absent. It burns with great heat and a long flame. It is of little metallurgical importance, as a good cannel coal, like anthracite, commands fancy prices for household purposes.

Subbituminous coal (sometimes called black lignite) is distinguished from true lignite by its color and freedom from apparent woody structure, and from bituminous coal by its quick slacking into thin plates on exposure to weather. In the West it is an important fuel.

Lignite, properly, refers to the distinctly brown coals and to those markedly woody or claylike in texture. Lignite as it comes from the mine carries 30 to 40 per cent moisture. It has no metallurgical importance except for burning for producer gas.

¹ Consulting engineer, New York.

SOME AMERICAN COAL ANALYSES

	H ₂ O	Volatiles combustible	Fixed carbon		B.t.u.
Anthracite:					
Anthracite region, Pa.....	5.7	86.2	5.9	0.6	13,830
Portsmouth district, R.I.....	2.6	65.3	18.9	0.3	9,310
Yampa field, Colo.....	3.4	75.6	14.0	0.6	11,740
Iron and Kane counties, Utah	4.4	58.0	29.4	2.3	8,910
Semianthracite:					
Russellville district, Ark.....	10.5	79.5	8.3	2.5	13,870
Sullivan County, Pa.....	9.3	75.6	11.7	0.8	13,120
Semibituminous:					
Tazewell County, Va.....	15.5	77.8	2.9	0.6	14,860
Cambria County, Pa.....	12.5	77.9	6.3	1.0	14,340
Coal Basin, Colo.....	19.4	70.8	7.4	0.5	14,880
Georges Creek, Md.....	16.6	72.1	10.7	1.3	12,380
Coal Hill, Ark.....	14.8	76.9	6.9	1.5	14,330
Iron and Kane counties, Utah	13.2	49.2	28.1	3.6	8,860
Bituminous:					
Pocahontas, W. Va.....	18.27	73.97	7.76	0.58	14,040
Pocahontas, W. Va.....	18.53	75.07	6.49	0.74	14,279
Pike County, Ky.....	36.2	56.7	4.3	1.4	14,180
Trinidad, Colo., non-coking, .	29.8	58.7	9.2	0.5	13,780
Stone Cañon, Cal.....	46.7	40.1	6.2	4.2	12,450
Saginaw district, Mich.....	31.5	49.8	6.8	1.2	11,780
Sangamon County, Ill.....	37.2	40.3	8.2	4.4	11,010
Esmeralda County, Nev.....	33.7	33.7	30.5	7.2	9,590
Subbituminous:					
Gallup district, N.M.....	36.5	43.3	7.5	0.7	11,230
Black Mesa, Ariz.....	32.6	46.9	10.6	1.1	10,500
Summit County, Utah.....	36.0	44.8	5.0	1.4	10,630
Mount Diablo, Cal.....	38.4	34.5	12.1	5.6	9,240
Glenrock district, Wyo.....	34.0	37.4	6.7	0.9	9,070
Coos Bay field, Ore.....	31.8	39.7	10.5	2.2	8,910
Colorado Springs field, Colo...	29.7	37.7	6.4	0.3	8,350
Miles City district, Mont....	26.2	35.4	9.2	0.8	7,670
Lignite:					
Goose Creek field, Ida.....	26.6	25.7	13.4	2.5	8,610
Missoula field, Mont.....	29.3	26.1	19.9	0.9	6,730
Camden district, Ark.....	26.5	24.4	9.7	0.5	6,360
Williams County, N.D.....	27.2	26.3	5.4	0.7	6,490
Harding County, S.D.....	24.0	24.3	10.2	0.6	5,650

The analyses in the accompanying table are not to be considered typical of the states represented, but are rather chosen to show the variations in chemical analysis and B.t.u. that may exist in the different classes. Analyses of coals from almost every district in the United States are found in U.S. Geological Survey *Professional Paper 48* and *Bulletins 232* and *290*, and in *U.S. Bureau of Mines Bulletin 5* (from which most of the accompanying analyses were taken) to which reference should be made when it is desired to know about what analysis coal from any given district may be expected to have.

Coke.—Coke is the chief metallurgical fuel. It may be defined as the coherent cellular residue of the destructive distillation of coal at high temperatures.¹ The

¹ R. in "American Fuels," edited by BACON and HAMOR, McGraw-Hill Book Company, Inc., New York, 1922.

nonferrous metallurgist need not concern himself with the methods of production, whether in beehive or by-product ovens, or with the type of the latter, so long as the coke is a satisfactory fuel.

Coke, even though produced at one time from the same coal, differs in appearance according to the portion of the oven that it occupies. In by-product coke the material that was originally adjacent to the wall of the oven can be distinguished by its cauliflower appearance and the dense layers of deposited carbon. The coke is classed according to the way it breaks, as "blocky" or "fingery." As a rule, the latter variety comes from coals carrying over 30 per cent volatile matter. The fingering is likely to be highly pronounced if the coal has a high oxygen content, but this condition can be controlled to some extent by the heat-treatment given.

In general, the cell structure of coke can best be examined by breaking a lump crosswise and noting the appearance of the fresh surfaces. The cells in those pieces that were nearer the wall are usually much larger than in pieces that were nearer the center of the oven. Also along any break the surface that was nearer the wall will have a granular appearance and well-defined cell openings. The surface that was nearer the center of the oven will have a graphitic luster, and the cell openings will be flatter and usually not so sharply defined. It is difficult to obtain a typical surface in a direction perpendicular to the wall of the oven except by grinding or sawing, as any break in the coke in this direction is almost always along a fracture, and the coke surface is obscured by deposited carbon.

The color of coke is relatively unimportant. If a large amount of water has been used in quenching, the coke will be dark, while a minimum amount usually gives a light-gray color. The silvery, glossy appearance formerly much sought after probably indicates a heavy deposit of carbon, favored by slow uniform evolution of a very rich gas.¹

Transverse, pebbly seams in the coke are due to the presence of small globules of pure carbon having no apparent cellular structure. These seams appear to be characteristic of the more oxygenated coals.

The surfaces always show some cracking, and if these cracks are present to excess the coke is of inferior grade.

The apparent specific gravity of coke is the specific gravity including the cell spaces. The true specific gravity is that of the coke substance proper. The porosity = $1 - \frac{\text{apparent specific gravity}}{\text{true specific gravity}}$. The apparent specific gravity is increased by coking under pressure and is decreased by rapid coking. According to Sperr,¹ the same coal will give cokes with apparent specific gravities running from 0.73 to 1.04, merely by varying these two factors.

Apart from proper burning qualities, the most important quality in coke is its crushing strength. Probably too much importance is ordinarily attached to this quality, however, as even in the large iron blast furnaces the coke does not have to withstand a crushing burden of over 50 lb. per sq. in.,² and a very poor coke will easily stand this load.

Fulton assumes the compressive strength at one-fourth the ultimate strength and gives a minimum result of 170 lb. compressive strength for a Pennsylvania beehive coke, from which his figures run up to 804 lb. on a Glassport by-product coke.

Resistance to impact and abrasion are much more important qualities from a practical standpoint than is crushing strength. The former is measured by the "shatter" test.

¹ *Ibid.*

² *U.S. Geol. Survey Bull.* 336.

Shatter Test for Coke.—As prescribed by the A.S.T.M. the apparatus consists essentially of a box capable of holding at least 100 lb. of coke, supported with the bottom 6 ft. above a cast-iron plate. The doors on the bottom are so hinged and latched that they will swing clearly away when open and will not impede the fall of the coke. Boards are placed around the cast-iron plate to prevent pieces of coke from being lost.

Each sample weighs approximately 50 lb. and is selected at random, by using a 2-in. tine fork. The sample is cool when tested but not artificially dry.

The entire sample, which has previously been weighed, is placed in the box, spread evenly over the bottom, and dropped on the cast-iron plate. The entire material is dropped four times. The small material including the dust is returned to the box with the large coke each time in order to represent as nearly as possible the practical conditions to which coke is subjected.

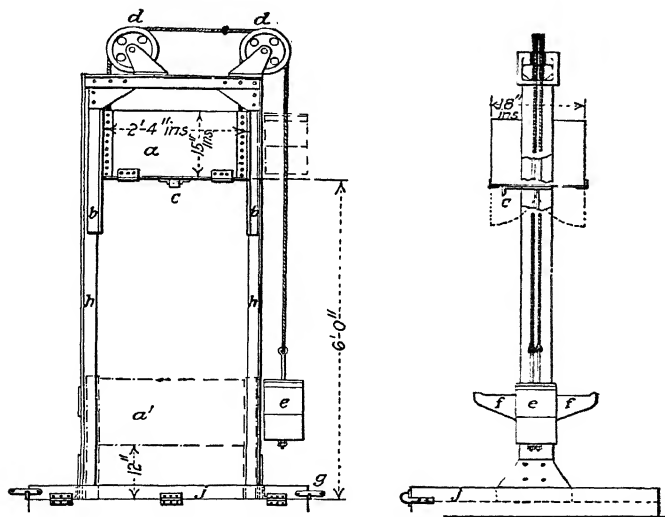


FIG. 1.—Shatter test apparatus for coke.

After the fourth drop, the coke is screened on a wire screen with square holes, 2 in. in the clear; the coke that remains on the screen and the portion that passes through are weighed, and the breakage is determined. If the sum of the weights indicates a loss of over 1 per cent, the test is rejected and a new one made.

A convenient type of shatter-test apparatus is shown in the illustration. The box is hung by cable wire over pulleys, which permit the box to be lowered for filling. This arrangement saves the labor and time that is involved in climbing up to fill the box before each drop.

A very good coke may be dropped 10 ft. onto a hard surface without shattering. There seems to be no reliable way of estimating abrasion.

A. C. Fieldner, of the Bureau of Mines, is of the opinion that much better results are obtained by using a series of screens, 2, 1 1/2, 1, and 1/2 in., rather than the single 2-in. screen now in use.

The specific heat of Saar coke is given by Simmersbach as follows:

Temperature, degrees Centigrade	Specific heat	Temperature, degrees Centigrade	Specific heat	Temperature, degrees Centigrade	Specific heat
100	0.1571	800	0.2927	1500	0.4283
200	0.1765	900	0.3121	1600	0.4477
300	0.1958	1000	0.3314	1700	0.4671
400	0.2152	1100	0.3508	1800	0.4864
500	0.2346	1200	0.3702	1900	0.5058
600	0.2540	1300	0.3896	2000	0.5252
700	0.2733	1400	0.4089		

But he gives for Ruhr coke a specific heat of 0.377 at 750°C.; 0.390 at 840°C.; 0.394 at 950°C.; and 0.400 at 1050°C. If there is so great a difference between Saar and Ruhr coke, it is doubtful whether these figures could be used for American practice with any more certainty than, for example, the mantissas of a number of logarithms.

The moisture, ash, volatile combustible, and fixed carbon will usually be determined. Moisture is an expense if it is paid for as coke, and incidentally it decreases the strength of the coke; ash also is not fuel, and, in addition, must be slagged and handled; volatile combustible promotes top fire in the blast furnace. Methods of making these determinations may be found in the manuals of assaying, or the *Proceedings* of the A.S.T.M.

A high ash in coke does not necessarily indicate weakness. Cokes from washed coals may be weaker than those from the same coals unwashed. Ordinarily coke fines will show a higher ash than the large pieces, mainly because the large pieces of slaty material cause breakage along the planes in which they occur, themselves break, and the fragments impoverish the fines.

Other chemical impurities often tested for are phosphorus and sulphur, neither of which need disturb the nonferrous metallurgist. The analysis of the ash also is ordinarily not a disturbing factor in nonferrous metallurgy, although it sometimes becomes of importance.

The superiority of coke as a metallurgical fuel lies in its hardness and cleanliness (absence of smoke coming under this latter category) and in its high calorific intensity. By this is meant the ability to burn a large amount of it in a small space in a short time, thus cutting down conduction losses and making it possible to obtain higher temperatures than with coal. This latter feature is especially important in metallurgy, as it is the last 100 or 200° that does the work, and not the several hundred before the fusion or reaction temperatures. Because of its higher kindling temperature, more of the fuel can reach the active zone in the blast furnace, hence giving freedom from top fire. To the ferrous metallurgist the higher ratio of carbon to sulphur is also an advantage; to the nonferrous worker, this point is not important. Coke is attacked appreciably by carbon dioxide at 500°C., and the reaction becomes rapid at 900°C.

Low-temperature Distillation.—This name is given to a number of processes involving a much higher by-product recovery through destructive distillation at much lower temperatures (not over 1000°F.) than those of the ordinary coking processes. The nonferrous metallurgist is not interested in the process itself, but in its product. The coke is said to be of good enough quality for metallurgical use, but the writer knows of no one who has so used it.

Wood.—In primitive countries, wood is used to some extent as a reverberatory and roaster fuel. It has the advantages of having a long flame, yielding a small quantity of ash, and being free from any harmful constituents. Roberts-Austen gives the following approximate composition for air-dried wood: C, 40; H, 4.8; O, 32.8; N, 0.8; ash, 1.6; and H_2O , 20 per cent. In the United States wood is frequently used in sawmills and woodworking plants, but not by metallurgical plants.

Wood is usually sold in the United States and the British Empire by the cord of 128 cu. ft. Green red oak is supposed to weight about 5800 lb. per cord, 3400 lb. per cord when air dry; air-dry ash, beech, birch, and sugar maple, 3800 to 4000; air-dry white oak, 4300; hickory, 4600; elm, 3000; and red maple, 3200 lb.

Green wood is about 57 per cent as effective as coal and about 75 per cent as effective as dried wood, pound for pound. (The wood substance has the same heating effect in both the dry and the moist wood, but 1 lb. of dry wood contains more wood substance than does 1 lb. of wet wood, apart from the difference in moisture to be evaporated.)

For a slow fire, a combination of wet and dry wood is oftener more satisfactory than is dry wood alone. It need hardly be said that in a wood-fired reverberatory the fire box must be much larger than in a coal-fired one. Air-dried wood has a heating value of about 3,470 cal. per g. The chemical constitution of dry cellulose is $(C_6H_{10}O_5)_x$.

Charcoal is also much used as a fuel in crucible melting, and to a greater extent as a cover in the crucibles, even when coke is the fuel, since there is so little ash from the charcoal to contaminate the metal. It is also used as a cover in copper-refining operations in reverberatories, although low-sulphur cokes have largely supplanted it there.

In the form of charcoal, wood is extensively used in iron smelting and, in primitive countries, in blast-furnace work of all sorts. Wherever it can be obtained, it is a welcome fuel in iron smelting because of its freedom from sulphur and phosphorus, and "charcoal iron," because of its purity, sells on a quality basis that enables this industry to compete with coke furnaces. To the nonferrous smelting industries, the sulphur and the phosphorus of coke are not objectionable so that charcoal sells on a heat-unit basis in competition with coke which renders charcoal out of the question in most regions.

Charcoal is an ideal fuel for gas producers because of the freedom from tarry matter and ash, but here again the price in civilized communities makes it uneconomic. A good charcoal will run about 93 per cent C, $1\frac{1}{2}$ per cent O, $2\frac{1}{2}$ per cent H, and 3 per cent ash. Its calorific power can be computed from its analysis, considering all the oxygen to be combined with hydrogen as water.

Peat.—The great objection to peat as a fuel is the large amount of moistures it contains. As it lies in the bog it carries about 85 per cent water, and to evaporate this requires about as much heat as that latent in the dry residue. It will air dry to a moisture content of 20 to 60 per cent, an average figure being about 35 per cent, and can then be gasified in a producer. In a Mond producer the contained moisture largely replaces steam in the air feed. The residual moisture after air drying is apparently held firmly with a decay product of the original vegetable material, which can be broken down at 150 to 200°C., after which the moisture can be expelled by pressing. However, in view of the difficulties of preparation and the low calorific value (2850 to 5200 cal. per g.), it is doubtful whether peat can be considered as having any field metallurgically except via the gas producer.

Oil.—Oil is widely used in reverberatory smelting, and, as a rule, its use is determined largely by competition in costs between coal and oil. The use of the latter is sometimes indicated even at noncompetitive prices, as in firing doré furnaces, or

crucible furnaces for gold and fine-silver melting, where its use saves the necessity of resmelting ashes, and also of watching the loading and tramping out of the ash cars to see that no small ingots go with them.

In making the above statement as to "competitive costs," it is realized that the costs of the heat units are not the whole story, as the matters of handling the fuels, the cost of ash removal, the advantage of quicker control of the fire with oil, lessened stand-by losses, etc., also enter.

In general, the calorific power of an oil can be computed from the formula

$$\text{B.t.u. per pound} = 18,650 + 40(B\text{é}^\circ - 10)$$

With oil fuel, because of the large amount of hydrogen present, a CO_2 content of 15 to 16 per cent in the waste gases would be the theoretical maximum, and an analysis of about 14.5 per cent CO_2 would indicate close to an operating maximum.

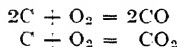
Combustion.—Before the subject of gas fuels can be taken up, it seems necessary to treat the general subject of combustion, since the manufacture of water gas and producer gas is an imperfect combustion.

For metallurgical purposes, combustion is the combination of a fuel with oxygen, the fuel being carbon, hydrogen, or a combination of carbon and hydrogen; or of carbon, hydrogen, and oxygen, or carbon monoxide. The oxygen is usually furnished by air, the composition of which is about:

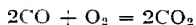
	By volume	By weight
Oxygen.....	20.91	23.15
Nitrogen.....	79.09	76.85

In the above table the rare gases are included with the nitrogen, as is also the carbon dioxide of the air, which amounts usually to about 0.04 per cent.

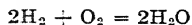
The most important point in the combustion of carbon is that it unites with oxygen to form two distinct compounds, carbon monoxide and carbon dioxide



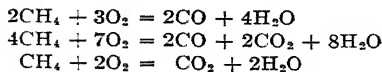
the first being formed when there is not enough oxygen to burn the carbon to the dioxide. The monoxide itself can burn when brought in contact with a further supply of air



Hydrogen always burns to water



hydrocarbons to a mixture of monoxide and water if only enough air is present to produce that result; to a mixture of monoxide, dioxide, and water, with more air, but still not enough for complete combustion; and to carbon dioxide and water if sufficient or an excess of air be present.



A carbohydrate, such as the cellulose of wood, gives the same end products as a hydrocarbon, the oxygen of the carbohydrate coming off as water and oxides of carbon. The partial oxidation already existing in the carbohydrates, of course, decreases their calorific power as compared with hydrocarbons.

Sulphur will usually burn to sulphur dioxide (SO_2)—under some conditions partly to sulphur trioxide (SO_3). This reaction is of little importance in power plants, but is of tremendous importance in roasting and smelting. The metal with which the sulphur is combined will be changed to an oxide, or may, in exceptional cases with a limited amount of air, remain behind as a lower sulphide.

The requisites for combustion are a fuel, a supporter of combustion (usually oxygen), and a high enough temperature to initiate the reaction. Exact kindling temperatures are difficult to determine, but some approximations are given here.

	IGNITION TEMPERATURE, DEGREES FAHRENHEIT
Sulphur.....	470
Charcoal.....	650
Bituminous coal.....	620
Anthracite.....	1290
Acetylene (C_2H_2).....	800
Ethane (C_2H_6).....	1000
Ethylene (C_2H_4).....	1020
Hydrogen (H_2).....	1095
Methane (CH_4).....	1290
Carbon monoxide (CO) moist.....	1200

The unit of heat used in English-speaking countries is the British thermal unit (B.t.u.). This is variously defined as the amount of heat required to raise the temperature by 1°F . of 1 lb. of water at or near its maximum density 39.1°F ., or as one-one hundred eightieth of the heat required to raise the temperature of 1 lb. of water from 32 to 212°F . For practical purposes, the definitions amount to the same thing.

For most scientific work, the c.g.s. unit is used. This is the calorie, the amount of heat necessary to raise the temperature of 1 g. of water from 10 to 1100°C . The calorie is sometimes also defined as the amount of heat necessary to raise the temperature from 0 to 1°C . and also from 4 to 5°C . The differences in these definitions are negligible in metallurgical work.¹ The large Calorie (kilogram calorie) is also sometimes used, the kilogram of water replacing the gram in the definition.

Different substances give out different amounts of heat when they combine, these amounts being known as their heats of combination. These are usually expressed in the calories given out (or absorbed, as the case may be) when a number of grams of the substances equal to their atomic weights react together. Thus, when it is said that the heat of combination of carbon burning to CO is 29,160 cal., it is meant that 12 g. of carbon and 16 g. of oxygen combine to form 28 g. of CO and give out 29,160 cal. in so doing.

It is obvious that the available heat of such a reaction depends on the condition of the end products. The heat of reaction of hydrogen burning to water, the water remaining a gas, is not the same as where the water remains as a liquid. For this reason combining heats must be used with a knowledge of what the state of the end products is, and, in general, it is unsafe to attempt to work with too high a degree of accuracy in these combination calculations. Therefore, if the calorific value of a fuel

¹ 1 B.t.u. = 778 ft.-lb.

1 cal. = 0.4265 kg.-m. (g. = 981).

crucible furnaces for gold and fine-silver melting, where its use saves the necessity of resmelting ashes, and also of watching the loading and tramping out of the ash cars to see that no small ingots go with them.

In making the above statement as to "competitive costs," it is realized that the costs of the heat units are not the whole story, as the matters of handling the fuels, the cost of ash removal, the advantage of quicker control of the fire with oil, lessened stand-by losses, etc., also enter.

In general, the calorific power of an oil can be computed from the formula

$$\text{B.t.u. per pound} = 18,650 + 40(\text{Bé}^\circ - 10)$$

With oil fuel, because of the large amount of hydrogen present, a CO_2 content of 15 to 16 per cent in the waste gases would be the theoretical maximum, and an analysis of about 14.5 per cent CO_2 would indicate close to an operating maximum.

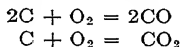
Combustion.—Before the subject of gas fuels can be taken up, it seems necessary to treat the general subject of combustion, since the manufacture of water gas and producer gas is an imperfect combustion.

For metallurgical purposes, combustion is the combination of a fuel with oxygen, the fuel being carbon, hydrogen, or a combination of carbon and hydrogen; or of carbon, hydrogen, and oxygen, or carbon monoxide. The oxygen is usually furnished by air, the composition of which is about:

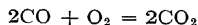
	By volume	By weight
Oxygen.....	20.91	23.15
Nitrogen.....	79.09	76.85

In the above table the rare gases are included with the nitrogen, as is also the carbon dioxide of the air, which amounts usually to about 0.04 per cent.

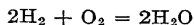
The most important point in the combustion of carbon is that it unites with oxygen to form two distinct compounds, carbon monoxide and carbon dioxide



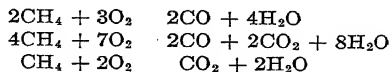
the first being formed when there is not enough oxygen to burn the carbon to the dioxide. The monoxide itself can burn when brought in contact with a further supply of air



Hydrogen always burns to water



hydrocarbons to a mixture of monoxide and water if only enough air is present to produce that result; to a mixture of monoxide, dioxide, and water, with more air, but still not enough for complete combustion; and to carbon dioxide and water if sufficient or an excess of air be present.



A carbohydrate, such as the cellulose of wood, gives the same end products as a hydrocarbon, the oxygen of the carbohydrate coming off as water and oxides of carbon. The partial oxidation already existing in the carbohydrates, of course, decreases their calorific power as compared with hydrocarbons.

Sulphur will usually burn to sulphur dioxide (SO_2)—under some conditions partly to sulphur trioxide (SO_3). This reaction is of little importance in power plants, but is of tremendous importance in roasting and smelting. The metal with which the sulphur is combined will be changed to an oxide, or may, in exceptional cases with a limited amount of air, remain behind as a lower sulphide.

The requisites for combustion are a fuel, a supporter of combustion (usually oxygen), and a high enough temperature to initiate the reaction. Exact kindling temperatures are difficult to determine, but some approximations are given here.

	IGNITION TEMPERATURE, DEGREES FAHRENHEIT
Sulphur.....	470
Charcoal.....	680
Bituminous coal.....	620
Anthracite.....	1290
Acetylene (C_2H_2).....	800
Ethane (C_2H_6).....	1000
Ethylene (C_2H_4).....	1020
Hydrogen (H_2).....	1095
Methane (CH_4).....	1290
Carbon monoxide (CO) moist.....	1200

The unit of heat used in English-speaking countries is the British thermal unit (B.t.u.). This is variously defined as the amount of heat required to raise the temperature by 1°F . of 1 lb. of water at or near its maximum density (39.1°F .), or as one-one hundred eightieth of the heat required to raise the temperature of 1 lb. of water from 32 to 212°F . For practical purposes, the definitions amount to the same thing.

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1 cal. = 0.4265 kg.-m. (g. = 981).

is calculated from its ultimate analysis, a check on this by actual experiment in a calorimeter is advisable if it can be obtained.

In most metallurgical operations, and in power plants, the end products are so warm that any water formed remains as a gas, and therefore throughout this book the lower values are used for the heats of combination of hydrocarbons and oxygen. The writer believes that the A.S.M.E. recommends the higher value (H_2O remaining as water) for calculations, but if the heat cannot be recovered the advantage in calculating on a purely theoretical basis is not apparent.

A change of state is always accompanied by the evolution or absorption of heat. Thus, 1 g. of ice melting to water at $0^\circ C$. absorbs 79.76 cal.; 1 g. of steam condensing to water at $100^\circ C$. gives out 538 cal.

The amount of heat (energy) required to dissociate a compound is the same as that evolved in forming the compound. And if a reaction occurs in two stages, the final result is the same as though it occurred in one.



Also of fundamental importance in calorific calculations is the fact that two distinct substances do not require the same amount of heat to raise their temperatures over the same range. The number of calories required to raise the temperature of 1 g. of a substance 1° from a given temperature is defined as its specific heat at that temperature. It usually varies slightly at different temperatures.

The specific heats of gases vary according to whether they are measured with the gas at constant volume or at constant pressure, the ordinary symbols for these being C_v and C_p .

The knowledge, then, of heats of combination and of change of state, specific heats, and temperatures at which certain dissociations and combinations begin are the essentials for the calculation of fuel problems, and also for many of those of drying, roasting, and smelting.

COMBUSTION DATA

Substance	Symbol	Atomic or molecular weight	C_p	C_v	Weight in pounds per cubic foot	B.t.u. per pound
Carbon.....	C	12.005	0.145	(0-100°)		
Hydrogen.....	H	1.008	3.4090	2.411	0.00561	4,380 ¹
Oxygen.....	O	16.00	0.2175	0.1723	0.08921	52,920
Sulphur.....	S	32.06	0.2346	(119-147°)		4,050 ²
Nitrogen.....	N	14.01	0.2350	0.1727	0.07808	
Carbon monoxide.....	CO	28.005	0.2425	0.1736	0.07806	4,350
Carbon dioxide.....	CO ₂	44.005	0.2010	0.1720	0.12341	
Methane.....	CH ₄	16.037	0.5929	0.486	0.04475	21,670
Acetylene.....	C ₂ H ₂	26.026	0.07361	21,020
Ethylene.....		28.042	0.404	0.07868	20,420
Ethane.....		30.058	0.08469	20,500
Sulphur dioxide.....	SO ₂	64.06	0.2175	0.1723	0.13272	945
Sulphur trioxide.....	SO ₃	80.06	0.22732	
Hydrogen sulphide.....	H ₂ S	34.076	0.245	0.09610	
Water vapor.....	H ₂ O	18.016	0.4805	0.3700		
Air.....			0.2417	0.1684	0.08072	

¹ Burning to CO; 14,540 burning to CO₂. ² To SO₂.

HEATS OF COMBINATION

	CALORIES
$C(12) + O(16) = CO$	29,160
$C(12) + 2O(32) = CO_2$	97,200
$CO(28) + O(16) = CO_2$	68,040
$2H + O = H_2O$	58,060
$S(32) + 2O(32) = SO_2$	69,260
$S(32) + 3O(48) = SO_3$	91,900
$C + 4H = CH_4$	22,250
$2C + 2H = C_2H_2$	-54,750
$2C + 6H = C_2H_6$	26,650
$2H + S = H_2S$	4,800

Maximum Theoretical Combustion Temperatures.—By making the somewhat violent assumptions that combustion is complete, but that no excess oxygen is supplied and that there is no radiation, a theoretical maximum temperature developed by any reaction can be computed. For combination with oxygen this would be as follows: if H is the heat of combination of m g. of the material under consideration, t_1 its initial temperature, t_2 its kindling temperature, C_{p1} its mean specific heat from t_1 to t_2 , n g. the mass of oxygen necessary for its complete combustion, t_3 its initial temperature, C_{p2} its mean specific heat from t_3 to t_2 , C_{p3} the mean specific heat of the products of combustion, and t_4 the final temperature, then

$$H = nC_{p2}(t_2 - t_3) + mC_{p1}(t_2 - t_1) + (m + n)(C_{p3})(t_4 - t_2)$$

Since t_4 is unknown and C_{p3} varies with it, it is necessary to assume a value of C_{p3} for about the temperature which it is believed will be reached, and calculate t_4 , then to correct the assumption by the result first obtained and recalculate the equation.

If air is used as the oxidizer, instead of oxygen, the equation is complicated by calculating the calories used in heating up the nitrogen of air to the final maximum

OXYGEN AND AIR REQUIRED FOR PERFECT COMBUSTION¹

1 kg.	Required kilograms		Product of combustion		Nitrogen in original air kilograms
	Oxygen	Dry air	Composition	Kilograms	
C.....	1.333	5.777	CO	2.333	4.444
C.....	2.667	11.555	CO ₂	3.667	8.888
CO.....	0.571	2.472	CO ₂	1.571	1.901
H.....	8.000	34.664	H ₂ O	9.000	26.664
CH ₄	4.000	17.332	CO ₂ , H ₂ O	2.750, 2.250	13.332
C ₂ H ₄	3.429	14.848	CO ₂ , H ₂ O	3.143, 1.286	11.419
Fe.....	0.286	1.238	FeO	1.286	0.952
Fe.....	0.429	1.857	Fe ₂ O ₃	1.439	1.428
Si.....	1.143	5.064	SiO ₂	2.143	3.921
P.....	1.290	5.586	P ₂ O ₅	2.290	4.296
Mn.....	0.291	1.221	MnO	1.291	0.969
S.....	1.000	4.333	SO ₂	2.000	3.333

¹ HOFMAN, "General Metallurgy."

calculated temperature. Obviously, actual working temperatures are always much less than maximum theoretical combustion temperatures, since there is always either an excess or a deficiency of oxygen (or air), and radiation and conduction cannot be guarded against.

If there is an excess of oxidizer, heat units are required to warm it to the final temperature; if there is a deficiency of oxidizer, not all the material is burned and the remainder must be heated, like so much inert material, or, for carbon, an inferior reaction takes place.

THEORETICAL MAXIMUM COMBUSTION TEMPERATURES¹

	DEGREES CENTIGRADE
Oxyhydrogen flame.....	3191
Hydrogen and dry air.....	2010
Hydrogen and dry air in 25 per cent excess.....	1764
Carbon monoxide with cold air.....	2050
CO and air, both at 700°C.....	2284
Natural gas and air.....	1806
Natural gas with air at 1000°C.....	2288
Coal gas.....	1875
Blue water gas.....	1960
Producer gas.....	1570
Thermit ($2\text{Al} + \text{Fe}_2\text{O}_3$).....	2694

¹ RICHARDS, J. W., "Metallurgical Calculations," vol. 1, pp. 36-39.

Combustion Losses.—It may be accepted as a general truth that any losses from soot in the products of combustion are negligible as compared with the unseen losses. When carbon burns to carbon monoxide, only one-third of its potential fuel value is realized. However black the smoke, it is seldom that the carbon thus lost amounts to 1 per cent of the weight of the coal, while 10 or 12 per cent of the weight of the coal may go off as CO without exciting any comment. This condition usually accompanies an undersupply of air and a smokeless chimney. The importance of checking fuel utilization under a boiler by analysis of the stack gases cannot, therefore, be overestimated.

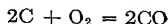
This is less true in metallurgical furnaces, where the character of the work performed in the furnace often dictates an uneconomical method of combustion.

Gas.—There is no question that in cleanliness, ease of control, and saving of labor gas is preeminent as a fuel. In the earlier days of metallurgy in the United States natural gas was quite a factor, but its importance has dwindled yearly with the exhaustion of wells, and the enforcing of prior claims of householders over factories in its use. It consisted, usually, of over 90 per cent methane and some hydrogen. Sexton¹ gives as a typical Ohio natural gas: H, 1.89; CH₄, 92.84; CO, 0.2; C₂H₆, 0.35; N, 3.82; and CO₂, 0.75 per cent. Natural gas has about 700 to 1200 B.t.u. per cu. ft.

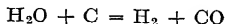
Therefore the gases that are of greatest metallurgical importance are manufactured. And as coal itself can be burned on surface grates or in powdered form so economically, artificial gas competes with coal only by virtue of local factors. Some of these are: (1) a summer supply of natural gas, so that the furnaces are equipped for gas burning—producer gas is then made during the winter months; (2) the gasification of low-grade fuels, lignite, colliery waste, etc.; (3) a shortage of water, making the use of internal-combustion engines almost a necessity; (4) a process that produces gas as a by-product, as the blast furnaces of the ironmakers, or nonferrous blast furnaces run under such peculiar local conditions as at Mansfeld.

¹ "Fuel and Refractory Materials," McGraw-Hill Book Company, Inc.

The theoretical basis of all processes for gas manufacturing is the burning of carbon to carbon monoxide in the presence of insufficient oxygen and the reduction of water vapor by heated carbon



and



Mond gas, probably the most widely known, carries about 14 to 16 per cent CO_2 , 10 to 12 per cent CO , 2 to 3 per cent CH_4 , and 25 to 29 per cent H_2 , the balance being nitrogen. The Mond process in its entirety contemplates the recovery of much of the nitrogen of the coal as ammonium sulphate, and a certain tar recovery. The process is essentially burning coal with a mixture of air and steam, though there are many refinements in a complete Mond plant, such as heat interchangers and scrubbers.

Where by-product tar and ammonia recovery is not intended, less steam is used and the gas ("Siemens gas") will contain about 25 to 28 per cent CO and 10 per cent H_2 , the B.t.u. with both Mond and Siemens gas being about 140 per cu. ft.

W. B. Chapman¹ gives as the average result of a number of well-run producer plants: CO_2 , 4.8; O_2 , 0.2; CO , 25.5; C_2H_4 , 0.4; H_2 , 12.1; CH_4 , 3.6; and N_2 , 53.4 per cent; B.t.u., 154.0 per cu. ft.; from coals averaging— H_2O , 3.5; per cent volatile, 32.8; fixed carbon, 56.2; ash, 7.5; S, 1.1; and total carbon, 74.8 per cent; B.t.u., 13,450. He states that the energy input to a producer is 97½ per cent in the coal and 2½ per cent in the steam of the output; the cold gas carries 74.5 per cent—11.5 per cent is in the sensible heat of the gas (usually wasted); 12 per cent loss due to radiation; 1 per cent loss as carbon in soot; 1 per cent loss to cooling water; and ½ per cent due to carbon in the ashes.

To cut down the loss as sensible heat in the gases and also to check radiation losses, the top of the fire bed should be kept as cool as possible, which means a thick fire bed. The bottom should be run as hot as possible. The even removal of ash from a producer is apparently extremely difficult, while it is at the same time the element that is chiefly conducive to efficiency.

While the chemistry of making producer gas is apparently simple, it is really a complex study in equilibrium data, and anyone making this gas should refer to a paper published by the Bureau of Mines, "Essential Factors in the Formation of Producer Gas," by Clement, Adams, and Haskins. The laboratory equilibrium data are not applicable because the gas goes through the fuel bed so rapidly that the time element is lacking.

Without enrichment of the gas, Mond gas² will give the following furnace temperatures:

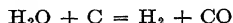
Without preheating air or gas.....	950–1150°C.
Continuous air recuperation.....	1150–1350°C.
Reversing air regeneration.....	1300–1400°C.
Reversing air and gas regeneration.....	1400–1800°C.

In Europe, lignite and peat are also used in Mond producers. Waldemar Dyrssen states that bituminous coal, treated properly, will give the following: C_2H_4 , 0.6; CH_4 , 3.6; CO , 29.1; H_2 , 13.3; CO_2 , 3.4; and N_2 , 50.0 per cent (no account taken of tarry vapors and H_2O). Water gas (blue gas) is the reaction product of steam and heated carbon.

¹ *Chem. & Met. Eng.*, Aug. 13, 1923.

² COFFIN, in "American Fuels," McGraw-Hill Book Company, Inc., 1922.

Theoretical "blue gas" would be 50 per cent H and 50 per cent CO produced by the reaction



Typical analyses, as actually made,¹ are:

	1	2
CO	43.5	39.1
H ₂	47.3	49.3
CH ₄	0.7	0.8
CO ₂	3.5	6.8
O ₂	0.6	0.3
N ₂	4.4	3.7
	<u>100.0</u>	<u>100.0</u>
B.t.u. per cubic foot	302	295

The reaction is exothermic, and consequently external heat is required to maintain it. The theoretical reaction temperature of blue gas is about 3500°F. against 2400 to 2900°F. for water gas. It is impossible to produce a smoky flame when firing with blue gas.

Pulverized Fuel.—Pulverized fuel has been used in metallurgical furnaces for about thirty years. There are, in general, two types of pulverized fuel plants as follows.

Storage System.—In this type of pulverized fuel plant, the coal is stored in bunkers over the pulverizers which are of the hammer-mill or ball-mill type. The coal is pulverized in such mills and usually conveyed by air through a cyclonic separator to the pulverized coal bunker. The fuel is then fed from the pulverized fuel bunker by a screw conveyer and air into the furnace. The advantages of this system are that coal may be pulverized at any time during the day. If sufficient pulverizing apparatus is installed, the pulverizing process can take place during the off-peak hours. The disadvantage of this type of system is that where the fuel is wet as introduced into the plant it must be dried, otherwise the moisture is carried through to the pulverized fuel bunker where it may cake. Another disadvantage in this type is that there is more opportunity for explosion of the mixture of fuel dust and air.

Unit System.—In this system, the fuel is stored in an overhead bunker where it is usually fed by gravity to a unit pulverizer which will be of the hammer-mill or ball-mill type. Here the fuel is pulverized and fed by air directly into the furnace. The advantages of this type of system are that the explosive hazard is less. A drying plant is seldom required, and the first cost can be less than the storage system. The disadvantage of this system is that fuel must be fed to the pulverizing unit and conveyed constantly from the pulverizing unit in very nearly the exact requirements of the furnace. Any partial stoppage in the fuel supply may result in inefficient operation of the furnace. Because of this possibility, frequently two pulverizing units are installed for each furnace so as to avoid any shutdown.

The general advantages of pulverized fuel as opposed to stoker-fired or hand-fired furnaces are as follows: (1) Pulverized fuel can be burned with nearly all the advantages of gas. Excellent control of temperatures and combustion in the furnace can be

¹ *Ibid.*, pp. 1005-1006.

secured with this mixture. (2) Lower grade fuels can be utilized in a pulverized form than can be utilized efficiently in other burning processes. Where coal with a very low fusing ash is used, care in the design of the furnace must be taken to avoid slagging.

To enable the pulverizing equipment to work at maximum efficiency it is usually necessary to dry the coal. Drying also allows the coal to flow more freely and increases the temperature of the flame. The usual method is to use a rotary drier, which should be fitted with apparatus for catching the dust and should be operated at such a low temperature that volatile-combustible matter will not be driven off.

If a hammer mill or disk grinder is used for pulverizing, a tramp-iron separator must be placed between the drier and the grinder.

Disk crushers, ring rolls, Raymond mills, and tube and conical mills are used for this crushing. Apart from the remarks in Chap. III on crushing, it may be noted that high-volatile coals crush more readily than do carbonized fuels or anthracite.

In utilizing pulverized fuel, it should be remembered that the choice of the furnace is highly important. There must be a spot in the furnace adjacent to the point of introduction of the mixed fuel and air which is kept sufficiently hot to keep the stream continuously ignited. The furnace must be so designed as to permit the creation of a long lazy flame without this flame impinging on any part of the refractory or any metal part of the furnace. Where low-fusing-point coal is used, the furnace must be deep enough to permit the falling particles of ash to cool below their slagging point before reaching the ash pit.

It should be noted that, in general, the cost of the furnace represents about half the total cost of a unit-type pulverizer plant. The power required to pulverize the coal and to conduct it by air into the furnace approximates 18 kw.-hr. per ton of coal fired. This amount of power compares very favorably with the amount required for the usual stoker-fired plant.

It should be noted, in figuring on gaseous fuels, that the B.t.u. per cubic foot of gas is not so important as is the net B.t.u. per cubic foot of products of combustion, and that on this figure depends the theoretical flame temperature. On this basis, blue water gas is a better fuel than is natural gas, although the latter contains nearly four times as many B.t.u. per cubic foot as does the former.

The following table is given by Carl J. Wright,¹ of the Combustion Utilities Corp.

	Gross B.t.u. per cubic foot	Net B.t.u. per cubic foot	Net B.t.u. per cubic foot of combustion products	Theoretical flame temperature, degrees Fahrenheit
Natural gas...	1,131	1,025	86.6	3325
Coal gas.....	636	570	89.5	3410
Blue water gas	310	285	102.0	3565
Producer gas..	157	148	73.0	2860

It is obvious from such a table that, on any process requiring temperatures of 2500° or over, the producer gas gives but a small percentage of its heat to the process and must be used with regeneration or waste-heat boilers to make it an economical fuel.

Surface Combustion.—The theory of surface combustion is to feed a mixture of a combustible gas and air into a refractory mass at a speed greater than the rate

¹ *Chem. & Met. Eng.*, Nov. 12, 1923.

of propagation of the flame. In the refractory mass, the gas mixture spreads out, burns, and heats the refractory to incandescence. The combustion is without visible flame. The danger is in a stoppage of the gas current, in which case back-firing will take place. The metallurgical application of the process is in crucible and muffle heating.¹

Fuel Novelties.—To the writer's knowledge, neither colloidal fuel nor Trent amalgam has any present metallurgical importance, but both fuels present some interesting possibilities.

Colloidal fuel was a discovery made during the first World War—that by means of a suitable peptizing agent a large amount of finely powdered coal or tar could be held in dispersion in fuel oil.² The coal is prepared for dispersion by being ground so that about 95 per cent passes 100 mesh and 85 per cent passes 200 mesh. The peptizing agent has not been made public. Since the coal is partly stored in the intermolecular spaces, the fuel is most economical of storage space and has an extremely high calorific value per unit volume. Its indicated use is, therefore, for naval purposes. The remarks concerning oil firing cover colloidal fuel. The process and the product are covered by U.S. patent 1447008 of Feb. 27, 1923, granted to Lindon W. Bates.

Because of the similarity of colloidal fuel to oil, there seems to be a possibility of obtaining the advantages of liquid fuel, where it is higher in price than coal, at a lower cost than straight oil firing will give, together with the possibility of great volume intensity for the heat.

Trent Process.—For this process, the coal must be pulverized to about 200 mesh, at which fineness most of the ash-forming material is free from the carbonaceous matter. A sludge of pulverized coal and water is then run into a tank, where oil is gradually added and the mixture rapidly agitated or stirred. There will come a point where the coal and the oil agglomerate into a mass, the so-called "amalgam," from which most of the ash and water are excluded. The water and ash are then drawn off and the amalgam washed. On exposure to the air the amalgam dries to 2 or 3 per cent of moisture. Either light or heavy mineral oils can be used, the proportion usually being about 1 of oil to 2 of coal. The oil can be distilled off at temperatures below which the hydrocarbons of the coal begin to come off, and the oil reused, while the coal is recovered as a dry powder, which is, because of its low ash, peculiarly valuable for coal-dust firing.³

Coal Storage.—To avoid fires from spontaneous combustion of coal, H. C. Porter of the Bureau of Mines gives the following rules: (1) Store in small piles, no part of which is more than 8 ft. from an air-cooled surface. (2) Store only screened coal, larger than 1 in. (3) Rehandle any part of a pile reaching 120°F. (4) Avoid through air circulation or ventilation in piles, bunkers, or bins. (5) Avoid freshly mined or crushed coal. (6) Keep the piles away from steam pipes and other external sources of heat. (7) Use the oldest coal first. (8) For large stocks, procure coal of low friability and keep breakage as low as possible.

The introduction of liquid CO₂ or of carbon tetrachloride fumes will often check the tendency to ignite. Storage under water is, of course, still better.

Rule 2 above is predicated on the greater surface in proportion to bulk afforded by the fine coal for oxidation. Some authorities, however, say the best results are obtained where sufficient fines are used to fill in between the lumps, thus avoiding

¹ See BONE, "Coal and Its Scientific Uses," Longmans, Green & Co.; HAMOR and BACON, "American Fuels," McGraw-Hill Book Company, Inc.; *Trans. A.I.M.E.*, Vol. 43, p. 612, for further particulars.

² Many of the particles are far above colloidal size, but are held suspended by the peptizing agent, despite their dimensions.

³ Cf. PERROTT and KINNEY, "Laboratory Studies of the Trent Process," *U.S. Bur. Mines Serial* 2263.

air circulation. In dumping from a trestle to the ground, there will be a certain segregation of the falling material, the coarse having a tendency to fall to the outside, the fines to the inside. Somewhere between will be a region where the coal is of intermediate size with a ventilating current just sufficient to obtain the maximum rise in temperature. The segregation of falling material of different sizes is the basis of the argument against fines, rather than the size itself.

Some authorities also consider that it is unnecessary to rehandle coal before it reaches 150 to 160°F., rather than the 120°F. given by Porter. It is obvious that coal storage should be so planned that no heat is added to it from the outside.

While the usual practice is to buy coal in summer to store for winter, the opposite procedure gives better results. Winter-stored coal goes into storage at a low temperature—a preventive of spontaneous combustion—and by the time summer temperatures are reached the coal has aged—also a preventive of spontaneous combustion. This is written, of course, merely with the prevention of spontaneous combustion in mind and not with smoothing off the labor curve in view.

Coals of different origin and quality should be segregated in storage. The coal piles should not be flanked with wood, and the size of the individual pile should be such as to admit of rehandling in the course of a few days with the ordinary plant facilities. The temperatures of the piles should be watched—a temperature inside the pile 20° above that of the air is a sign for watchfulness, and if a temperature of 160°F. is reached the piles should be moved.

CHAPTER IX

THERMOMETRY AND PYROMETRY

By W. F. ROESER¹ AND M. S. VAN DUSEN²

Ideal Temperature Scale.—The ideal or fundamental scale is known as the “thermodynamic scale,” which has been defined by Kelvin to be such that “the absolute values of two temperatures are to one another in the proportion of the heat taken in to the heat rejected in a reversible thermodynamic engine working with a source and refrigerator at the higher and lower of the temperatures respectively.” The size or the magnitude of a unit degree of temperature is not a part of this definition. The unit degree of the centigrade scale is the one one-hundredth part of the temperature interval from the freezing point to the boiling point of pure water under standard conditions (760 mm. Hg pressure). This unit is not related in a rational way to any other physical quantity, but is purely arbitrary.

For converting temperatures from Fahrenheit to centigrade, or vice versa, we have $\frac{5}{9}(F. - 32) = C.$; $\frac{9}{5}C. + 32 = F.$

No thermometer or pyrometer makes use of the fundamental definition of the thermodynamic scale, *i.e.*, by measurements of heat transfer in a reversible thermodynamic engine. However the International Temperature Scale [1],³ unanimously adopted in 1927 by the Seventh General Conference of Weights and Measures (representing 31 nations), is based on a number of fixed points whose location on the Kelvin scale relative to the ice point has been determined by gas thermometers. Part I of the text of the scale, given below, is self-explanatory. A few editorial corrections, not affecting the scale itself, were made in 1933.

DEFINITION OF THE INTERNATIONAL TEMPERATURE SCALE

1. The thermodynamic centigrade scale, on which the temperature of melting ice and the temperature of condensing water vapor, both under the pressure of one standard atmosphere, are numbered 0 and 100°, respectively, is recognized as the fundamental scale to which all temperature measurements should ultimately be referable.

2. The experimental difficulties incident to the practical realization of the thermodynamic scale have made it expedient to adopt for international use a practical scale designated as the International Temperature Scale. This scale conforms with the thermodynamic scale as closely as is possible with present knowledge. It is designed to be definite, conveniently and accurately reproducible, and to provide means for uniquely determining any temperature within the range of the scale, thus promoting uniformity in numerical statements of temperature.

3. Temperatures on the international scale will ordinarily be designated as “°C.”, but may be designated as “°C. (Int.)” if it is desired to emphasize the fact that this scale is being used.

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² Physicist, National Bureau of Standards.

³ Numbers in brackets refer to references at end of chapter.

4. The International Temperature Scale is based upon a number of fixed and reproducible equilibrium temperatures to which numerical values are assigned, and upon the indications of interpolation instruments calibrated according to a specified procedure at the fixed temperatures.

5. The basic fixed points and the numerical values assigned to them for the pressure of one standard atmosphere are given in the following table, together with formulas that represent the temperature t_p as a function of vapor pressure p over the range 680 to 780 mm. of mercury.

6. Basic fixed points of the International Temperature Scale.

	°C.
a. Temperature of equilibrium between liquid and gaseous oxygen at the pressure of one standard atmosphere (oxygen point) $t_p = t_{760} + 0.0126(p - 760) - 0.0000065(p - 760)^2$	-182.97
b. Temperature of equilibrium between ice and air-saturated water at normal atmospheric pressure (ice point).....	0.000
c. Temperature of equilibrium between liquid water and its vapor at the pressure of one standard atmosphere (steam point) $t_p = t_{760} + 0.0367(p - 760) - 0.000023(p - 760)^2$	100.000
d. Temperature of equilibrium between liquid sulphur and its vapor at the pressure of one standard atmosphere (sulphur point) $t_p = t_{760} + 0.0909(p - 760) - 0.000048(p - 760)^2$	444.60
e. Temperature of equilibrium between solid silver and liquid silver at normal atmospheric pressure (silver point).....	960.5
f. Temperature of equilibrium between solid gold and liquid gold at normal atmospheric pressure (gold point).....	1063

Standard atmospheric pressure is defined as the pressure due to a column of mercury 760 mm. high, having a density of 13.5951 g. per cc., subject to a gravitational acceleration of 980.665 cm. per sec.², and is equal to 1,013,250 dynes per sq. cm.

It is an essential feature of a practical scale of temperature that definite numerical values shall be assigned to such fixed points as are chosen. It should be noted, however, that the last decimal place given for each of the values in the table is significant only as regards the degree of reproducibility of that fixed point on the International Temperature Scale. It is not to be understood that the values are necessarily known on the Thermodynamic Centigrade Scale to the corresponding degree of accuracy.

7. The means available for interpolation lead to a division of the scale into four parts.

a. From the ice point to 660°C. the temperature t is deduced from the resistance R_t of a standard platinum resistance thermometer by means of the formula

$$R_t = R_0(1 + At + Bt^2)$$

The constants R_0 , A , and B of this formula are to be determined by calibration at the ice, steam, and sulphur points, respectively.

The purity and physical condition of the platinum of which the thermometer is made should be such that the ratio R_t/R_0 shall not be less than 1.390 for $t = 100^\circ\text{C}$. and 2.645 for $t = 444.6^\circ\text{C}$.

b. From -190°C . to the ice point, the temperature t is deduced from the resistance R_t of a standard platinum resistance thermometer by means of the formula

$$R_t = R_0(1 + At + Bt^2 + C(t - 100)t^3)$$

The constants R_0 , A , and B are to be determined as specified above, and the additional constant C is determined by calibration at the oxygen point.

TABLE 1.—FIXED POINTS AVAILABLE FOR CALIBRATING THERMOMETERS AND PYROMETERS

Thermometric fixed point	Values on the International Temperature Scale				Temperature of equilibrium (t_p) in °C. as a function of the pressure ² (p) between 680 and 780 mm. Hg
	Assigned (primary points)		Determined (secondary points)		
	°C.	°F.	°C.	°F.	
Boiling point of oxygen.....	-182.97	-297.35	$t_p = t_{180} + 0.0126(p - 760) - 0.0000065(p - 760)^2$
Sublimation point of carbon dioxide.....	-78.5	-109.3	$t_p = t_{180} + 0.1443(t_p + 273.2) \log \left(\frac{p}{760} \right)$
Freezing point of mercury.....	-38.87	-37.97	
Melting point of ice.....	0.000	32.000	
Boiling point of water.....	100.000	212.000	
Boiling point of naphthalene.....	217.96	424.33	$t_p = t_{180} + 0.0367(p - 760) - 0.000023(p - 760)^2$
Freezing point of tin ¹	231.9	449.4	
Boiling point of benzophenone.....	305.9	582.6	$t_p = t_{180} + 0.194(t_p + 273.2) \log \left(\frac{p}{760} \right)$
Freezing point of cadmium.....	320.9	609.6	
Freezing point of lead ¹	327.35	621.23	
Freezing point of zinc ¹	419.48	787.06	
Boiling point of sulphur.....	444.60	832.28	
Freezing point of antimony.....	630.5	1,166.9	$t_p = t_{180} + 0.0909(p - 760) - 0.000048(p - 760)^2$
Freezing point of aluminum ¹	660.15	1,220.27	
Freezing point of Cu-Ag eutectic alloy ²	778.8	1,433.8	
Freezing point of silver.....	960.5	1,760.9	
Freezing point of gold.....	1,063.0	1,945.4	
Freezing point of copper ¹	
Melting point of palladium.....	1,083.2	1,981.8	
Melting point of platinum.....	1,555	2,831	
	1,773	3,223	

¹ Standard samples of these materials are procurable from the National Bureau of Standards with certificates giving the freezing point of the particular lot of metal. The values given in this table for these materials apply for the standard samples that are being issued as of the present date.

² 28.1 per cent copper and 71.9 per cent silver by weight.

The standard thermometer for use below 0°C. must, in addition, have a ratio R_t/R_0 less than 0.250 for $t = -183^\circ\text{C}$.

c. From 660°C. to the gold point, the temperature t is deduced from the electromotive force e of a standard platinum vs. platinum-rhodium thermocouple, one junction of which is kept at a constant temperature of 0°C. while the other is at the temperature t defined by the formula

$$e = a + bt + ct^2$$

The constants a , b , and c are to be determined by calibration at the freezing point of antimony, and at the silver and gold points.

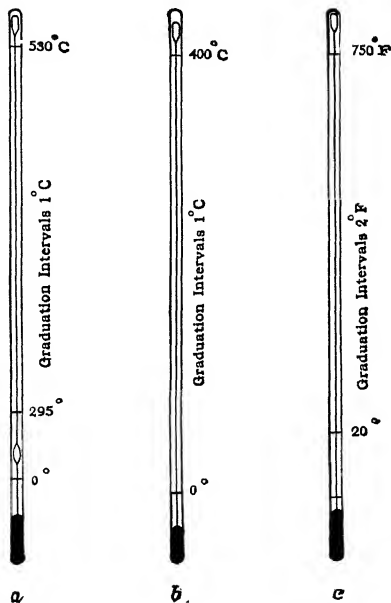


FIG. 1.—Etched-stem thermometers

d. Above the gold point, the temperature t is determined by means of the ratio of the intensity J_2 of monochromatic visible radiation of wave length λ cm., emitted by a black body at the temperature t_2 , to the intensity J_1 of radiation of the same wave length emitted by a black body at the gold point, by means of the formula

$$\log_e \frac{J_2}{J_1} = \frac{C}{\lambda} \left(\frac{1}{1336} - \frac{1}{(t + 273)} \right)$$

The constant C_2 is taken as 1.432 cm. degrees. The equation is valid if $\lambda(t + 273)$ is less than 0.3 cm. degrees.

A list of thermometric fixed points for which values have been assigned in the International Temperature Scale or determined accurately on that scale and which have been found useful in the calibration of thermometers and pyrometers is given in Table 1.

Mechanical Thermometry.—Table 2 gives a classification of thermometers of this type. In general, such instruments are not suitable for temperatures much above 500°C.

TABLE 2.—CLASSIFICATION OF THERMOMETERS

Mercury in glass (mercurial) . . .	{ Laboratory or chemical . . .	{ Etched or engraved stem
	{ Industrial	{ Enclosed scale (einschluss)
Indicating and recording	{ Pressure	{ Vapor-pressure
	{ Bimetallic	{ Liquid-filled
		{ Gas-filled

High-range Mercurial Thermometers.[2]—Figure 1 shows some characteristic types of etched-stem thermometers which are more extensively used than the enclosed-scale type. Type *a* (Fig. 1) is used at the National Bureau of Standards as a standard for comparison in the range 300 to 530°C. Type *b* is a continuous-scale instrument used in distillation tests and graduated in 1° intervals in the range 0 to 400°C. for total immersion, but actually used at about 3-in. immersion. Type *c* is graduated in 2° intervals for partial immersion and is used in oil testing and other industrial work.

In all such thermometers the scale should include the ice- or steam-point graduation (0 or 100°C. or 32 or 212°F.) which is used for checking purposes. The best made thermometers may not remain constant through repeated heatings, and the change may be allowed for by observations at the ice point.

The standard type *a* should not be used for partial immersion with the secondary bulb in the stem projecting into the air. This will result in large and uncertain errors.

Table 3, taken from *Bureau of Standards Circular 8*, 4th ed., gives some characteristics of the better grades of laboratory thermometers. Some grades of thermometers might have smaller tolerances than those indicated.

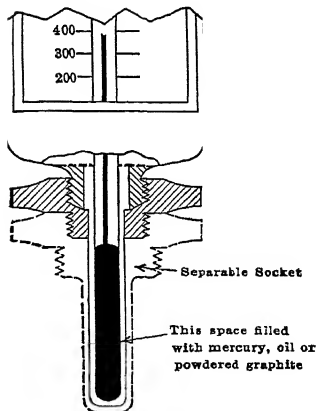


FIG. 2.—Industrial thermometer.

through this material, and the exterior conditions prevailing around the projecting stem. Such thermometers must be standardized or calibrated under the same conditions as when in use.

Emergent-stem Correction for Mercury-in-glass Thermometers.—Unless the thermometer is distinctly marked otherwise, it is to be assumed that it was graduated and calibrated for the condition of total immersion, *i.e.*, with bulb and all that part of the stem containing mercury at a uniform temperature. In many cases, however,

Unprotected glass thermometers are too frail for plant and shop and with the exception of those having lens fronts are comparatively difficult to read. Figure 2 illustrates the general type of industrial thermometer, which is mounted in a metal protecting tube with a projecting back. It has large distinct figures and scale marks and is provided with a threaded connection for installing the thermometer in pipes, boilers, etc. This type of thermometer has a considerable "lag," *i.e.*, its bulb does not rapidly follow the temperatures of the medium into which the thermometer bulb is immersed. Unless the mounting is so made that the bulb can be immersed to a sufficient depth, errors may arise that differ according to the construction of the protecting case, the material surrounding the bulb, the rate of flow of heat

TABLE 3.—TOLERANCES FOR MERCURIAL THERMOMETERS
CENTIGRADE TOTAL-IMMERSION LABORATORY THERMOMETERS

Temperature range, deg.	Graduation interval, deg.	Tolerance, deg.	Accuracy, deg.	Corrections stated to
Thermometers Not Graduated above 150°				
0 up to 150.....	1 or 0.5	0.5	0.1 -0.2	0.1
0 up to 150.....	0.2	0.4	0.02-0.05	0.02
0 up to 100.....	0.1	0.3	0.01-0.03	0.01
Thermometers Not Graduated above 300°				
0 up to 100	1 or 0.5	{ 0.5 1.0	0.1 -0.2	0.1
Above 100 up to 300			0.2 -0.3	0.1
0 up to 100	0.2	{ 0.4 0.5	0.02-0.05	0.02
Above 100 up to 200			0.05-0.1	0.02
Thermometers Graduated above 300°				
0 up to 300	2	{ 2 4	0.2 -0.5	0.2
Above 300 up to 500			0.5 -1.0	0.2
0 up to 300	1 or 0.5	{ 2 4	0.1 -0.5	0.1
Above 300 up to 500			0.2 -0.5	0.1
FAHRENHEIT TOTAL-IMMERSION LABORATORY THERMOMETERS				
Thermometers Not Graduated above 300°				
32 up to 300.....	2	1	0.2 -0.5	0.2
32 up to 300.....	1 or 0.5	1	0.1 -0.2	0.1
32 up to 212.....	0.2 or 0.1	0.5	0.02-0.05	0.02
Thermometers Not Graduated above 600°				
32 up to 212	2 or 1	{ 1 2	0.2 -0.5	0.2
Above 212 up to 600			0.5	0.2
Thermometers Graduated above 600°				
32 up to 600	5	{ 4 7	0.5 -1.0	0.5
Above 600 up to 950			1 -2	0.5
32 up to 600	2 or 1	{ 3 6	0.2 -1.0	0.2
Above 600 up to 950			0.5 -1.0	0.2
CENTIGRADE PARTIAL-IMMERSION THERMOMETERS				
Thermometers Not Graduated above 150°				
0 up to 150.....	1 or 0.5	1.0	0.1 -0.5	0.1

TABLE 3.—TOLERANCES FOR MERCURIAL THERMOMETERS.—(Continued)

Temperature range, deg.	Graduation interval, deg.	Tolerance, deg.	Accuracy, deg.	Corrections stated to
Thermometers Not Graduated above 300°				
0 up to 100.....	1	1.0	0.1 -0.3	0.1
Above 100 up to 300.....	1	1.5	0.5 -1.0	0.2
Thermometers Graduated above 300°				
0 up to 300.....	2 or 1	2.5	0.5 -1	0.5
Above 300 up to 500.....	2 or 1	5	1 -2	0.5
FAHRENHEIT PARTIAL-IMMERSION THERMOMETERS				
Thermometers Not Graduated above 300°				
32 up to 300.....	2 or 1	2	0.2 -1.0	0.2
Thermometers Not Graduated above 600°				
32 up to 212.....	2 or 1	2	0.2 -0.5	0.2
Above 212 up to 600.....	2 or 1	3	1 - 2	0.5
Thermometers Graduated above 600°				
32 up to 600.....	5 or 2	5.0	1 -2	1
Above 600 up to 950.....	5 or 2	10	2 -3	1

thermometers are used with part of the mercury in the stem extending outside the bath in which the bulb is immersed. In such cases, an emergent-stem correction should be applied, which can be calculated from the equation

$$\text{Correction} = 0.00016N(T - t) \text{ for centigrade thermometers}$$

$$\text{Correction} = 0.00009N(T - t) \text{ for Fahrenheit thermometers}$$

in which 0.00016 or 0.00009 is the apparent coefficient of expansion of mercury in glass, N is the number of degrees emergent from the bath, T is the bath temperature, and t is the average temperature of the emergent column. It will be noted that the sign of the correction is found from the formula; *e.g.*, if a relatively high temperature is being measured, $T - t$ will be positive, and the correction will be positive, indicating that the observed reading is too low and should be increased to the amount of the calculated correction.

Partial-immersion Thermometers.—To avoid the necessity of applying a correction for the emergent stem, so-called “partial-immersion thermometers” are extensively used, which are pointed and calibrated to read, as nearly as possible, correct temperatures when immersed to a definite mark on the scale, *e.g.*, 3 in. above the bottom of the bulb. The indications of such thermometers are obviously influenced to some extent by the temperatures outside the bath.

Such thermometers should be marked “. . . in. immersion” or its equivalent, and should be provided with a mark on the stem to indicate the depth of immersion.

The reliability of corrections determined for partial-immersion thermometers is necessarily somewhat less than that of the corrections for total-immersion thermometers, but this does not by any means imply that, if both thermometers are used with partial immersion, more accurate results will necessarily be obtained with the total-immersion thermometer.

For general laboratory use, the partial-immersion thermometer has some very evident advantages. In choosing the type to be preferred for any one kind of measurement, it is necessary to decide whether the possible errors incident to the use of a partial-immersion thermometer are larger than are permissible, and whether it is worth while to use a total-immersion thermometer under conditions approximating total immersion or, if the total-immersion thermometer must be used with a considerable portion of the stem emergent, to make an accurate determination of the stem correction. The magnitude of the possible error due to the use of a partial-immersion thermometer is best illustrated by an example.

Suppose a partial-immersion thermometer to have been standardized in a certain type of bath so that for a bath temperature of 300°C. the average temperature of the 300°C. length of emergent stem was 40° and that it is later used to measure the temperature of another bath at 300°. Under the most markedly different conditions, the average stem temperature could hardly differ by more than 50° from that which prevailed during the standardization of the thermometer. For this possible difference in mean stem temperatures in the two cases, the resulting difference in the indications of the thermometer (error as used) would be

$$0.00016 \times 300(50) = 2.4^{\circ}$$

In computing this error for a Fahrenheit thermometer, Fahrenheit temperatures are used, and 0.00009 is substituted for 0.00016. Except under very unusual conditions, the error under consideration would be hardly more than half that calculated above, or, in round numbers, about 1°.

Consider next the accuracy attainable by the use of a total-immersion thermometer likewise used with 300° of the mercury column emergent from the bath. If the average stem is actually 40° as before, the total stem correction is

$$0.00016 \times 300(300 - 40) = 12.5^{\circ}$$

It is at once evident that totally neglecting this stem correction will introduce an error many times as large as could possibly result from the use of a partial-immersion thermometer. If, on the other hand, the necessary care is taken to determine accurately the large stem correction, under the above conditions of use of the total-immersion thermometer, this stem correction could be determined to an accuracy of at least 0.5°, corresponding to an accuracy of about 10° in determining the average stem temperature, and in that case a somewhat higher accuracy could be attained with the use of a total-immersion thermometer. If, however, the stem corrections were determined by hanging an auxiliary thermometer beside the stem, the reading of this thermometer might differ considerably more than 10° from the average temperature of the stem, and the resulting error in the determination of the stem correction might exceed 1°, which is comparable with the error incident to the use of a partial-immersion thermometer. At lower temperatures, the case is slightly less favorable to the partial immersion thermometer, because a large part of the possible error in its use is due to differences in the temperature of the laboratory in different seasons.

Sources of Error in High-temperature Mercurial Thermometers.—Aside from the error that may be introduced in the use of high-temperature mercurial thermometers by a failure to observe the proper condition of immersion, there are two common

sources of error that can be attributed to actual defects in the manufacture of the instrument. These are (1) insufficient pressure above the mercury to prevent a separation of the mercury column, and (2) improper or insufficient annealing for use at high temperature. This latter defect may result in a rise of the reading with continued heating amounting to as much as $20^{\circ}\text{C}.$, in extreme cases.

In discussing the question of filling under pressure, it may be noted that experiments have shown that even for use as low as $150^{\circ}\text{C}.$ the filling under pressure of that part of the capillary above the column with an inert gas such as nitrogen is desirable. The pressure that must exist above the surface of the mercury to prevent separation of the mercury column varies with the temperature. The required pressure may be anywhere from 1 to 20 atm.

The failure to fill high-temperature thermometers under the proper pressure is often the cause of large errors. This defect can be detected sometimes by the broken appearance of the mercury column, but often the column breaks in a part of the stem which is not visible, and the defect is not detected. Drops of Hg distilled into the top are not a sure or even a good indication of insufficient pressure.

Improper annealing can be detected only by checking the indications of the instruments from time to time. Laboratories usually have facilities for checking the reading either in melting ice or in steam or perhaps comparing the thermometers from time to time with a thermometer that has an ice point on its scale. Industrial thermometers should also be tested for this source of error, although for this type annealing changes due to heating are not likely to be of importance below $300^{\circ}\text{C}.$

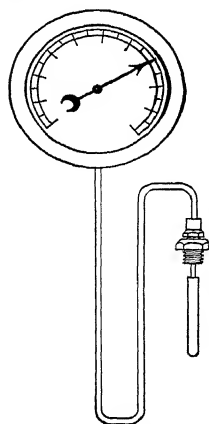


FIG. 3.—Distance-reading thermometer.

Indicating and Recording Thermometers.—Dial-type indicating thermometers may or may not be distance reading, *i.e.*, so constructed as to allow the indicator to be placed at some distance from the bulb. A distance-reading indicating thermometer is shown in Fig. 3. A recording thermometer employs a mechanism for making a continuous record of temperatures on a suitable chart. Indicating and recording thermometers may be divided into three general classes: electrical, pressure, and bimetallic. Electrical thermometers will be discussed in later sections (Thermocouples, Resistance Thermometers).

Bimetallic thermometers utilize the differential thermal expansion of two different metals suitably joined. The temperature-sensitive element of a bimetallic thermometer consists of two strips of metal having different coefficients of expansion, welded or brazed together all along their length. If such a composite strip is wound into a helix, changes in temperature will produce a turning moment which can be communicated to a dial by appropriate mechanism. Numerous other thermometric devices depending on the thermal expansion of solids have been devised, but have not come into general use.

Pressure thermometers comprise a bulb containing a liquid or gas, or both, connected by means of capillary tubing to some form of pressure gauge. The pressure under which a vapor-pressure thermometer operates is a function of the temperature of the bulb only, provided the free surface of the liquid is always in the bulb. The liquid selected to cover a given temperature range should have sufficient pressure at the lower temperatures in order to avoid the necessity of using delicate pressure gauges and mechanisms to indicate temperatures. Vapor-pressure thermometers are not well adapted to cover large temperature ranges on a single instrument, since the scale

will be very much compressed in the lower part of the range. The maximum temperature to which the liquid used in a vapor-pressure thermometer will be heated should not exceed its critical temperature. In general, the useful range for a given

TABLE 4.—CHARACTERISTICS OF PRESSURE THERMOMETERS

Designation	Material available for filling	Approximate temperature range, °C.	Scale
Vapor-pressure	{ Ethyl ether	40–190	Unevenly divided
	{ Ethyl alcohol	80–240	Unevenly divided
	{ Benzene	80–280	Unevenly divided
	{ Aniline	200–425	Unevenly divided
Liquid-filled...	{ Alcohol	Up–200	Evenly divided
	{ Mercury	Up–550	Evenly divided
Gas-filled.....	Nitrogen	Up–425	Evenly divided

liquid is included between its boiling point at atmospheric pressure and its critical temperature.

The initial pressure and the pressure range of a liquid- or gas-filled thermometer are adapted to the characteristics of the gauge used and the temperature range. Since the increase in pressure with temperature is approximately the same over the entire scale, the temperature scale is evenly divided. The pressures in liquid- or gas-filled thermometers are made relatively high to ensure a rugged gauge mechanism. Table 4 gives some characteristics of vapor-pressure and liquid- or gas-filled industrial thermometers.

The action of the vapor-pressure thermometer depends on the fact that the pressure inside the thermometer is determined solely by the temperature of the free surface of the liquid. It follows, therefore, that the thermometer must be so constructed that one free surface is always in the bulb. If this condition is fulfilled, the reading of the instrument will not be sensibly affected by changes in the temperature of the gauge and capillary, but it may be affected by the hydrostatic pressure of the liquid in the capillary unless all parts are at the same level.

This is a decided advantage over other types, if the connecting tubing and gauge are both to be subjected to considerable changes in temperature. On the other hand, if the vapor-pressure thermometer is not filled properly, *i.e.*, if the proportion of liquid is too great or too small as compared with the volume of the bulb, capillary, and gauge, large and uncertain errors may be introduced. As an example, take the condition met with in the use of high-temperature vapor-pressure thermometers.

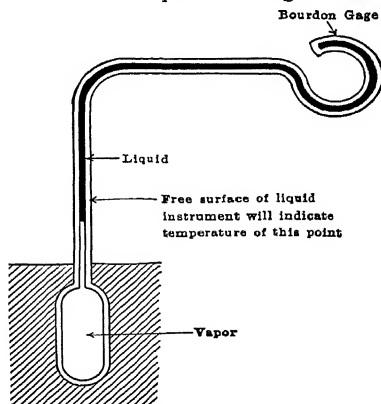


FIG. 4.—A defective vapor-pressure thermometer.

The bulb is usually much hotter than the capillary, so if the liquid only partly fills the capillary, and there is no liquid in the bulb, the capillary will contain the free surface of the liquid and the temperature indicated will be that of the portion of the capillary containing the free surface of the liquid as shown in Fig. 4. This temperature, which may be several hundred degrees lower than that of the bulb, will be indicated instead of the true temperature of the bulb.

Gas- and liquid-filled thermometers are entirely filled with either the liquid or the gas. The expansion of the liquid or gas in the bulb is transmitted through capillary tubing to a pressure gauge. These instruments are subject to error if the gauge and capillary are heated or cooled to temperatures differing from those under which they were calibrated. This error may be made negligible in many instances by reducing the volume of the capillary and gauge as compared with that of the bulb, or by using a compensator in the gauge and capillary.

On account of the mechanical construction of pressure thermometers, with regard to both the mechanism for indicating and recording and the necessity for using comparatively large and heavy bulbs, the accuracy that can be secured with this type of thermometer is lower than that obtained with mercury-in-glass thermometers. As with industrial thermometers, the conditions under which the instruments are originally calibrated must agree with those of use if errors are to be avoided with certainty. The greatest error is introduced, perhaps, when such instruments are calibrated in a liquid and are used to measure the temperature of a gas.

Thermoelectric Thermometers.¹—A thermocouple is fundamentally a pair of dissimilar electrical conductors joined together at one end. The measurement of temperature by means of a thermocouple depends upon the fact that a thermal e.m.f. is developed in a thermocouple when the junction of the two wires is at a temperature different from that of the other ends. In any given circuit containing any number of dissimilar metals, the algebraic sum of the thermoelectromotive forces generated is a function only of the temperatures of the junctions. Consequently, if all but one of the junctions of such a given circuit are maintained at some constant reference temperature, the e.m.f. developed in the circuit is a function of the temperature of that junction only. Therefore, by the proper calibration of such a device, it may be used to measure temperatures.

A thermoelectric thermometer is a device for measuring temperatures by utilizing the thermoelectric effects. In its simplest form, it consists of a thermocouple of two dissimilar metals that develop an e.m.f. when the junctions are at different temperatures and an instrument for measuring the e.m.f. developed by the thermocouple, connected together as shown in Fig. 5.

As long as the instrument is at essentially a uniform temperature, all the junctions in the instrument including the terminals will be at the same temperature, and the resultant thermal e.m.f. developed in the circuit is not modified by including the instrument. If the reference junction is maintained at some reference temperature, such as 0°C., the e.m.f. developed by the thermocouple can be determined as a function of the temperature of the measuring junction. The device can then be used for measuring temperatures. It is not necessary to maintain the reference junction at the same temperature during use as during calibration. However, the temperature of the reference junction in each case must be known. For example, let the curve in Fig. 6 be the relation between the e.m.f. E and temperature t for a particular thermocouple with the reference junction at 0°C. Suppose the device is used to measure some temperature and an e.m.f. E_x is observed when the reference junction is

¹ The major part of the discussion of thermocouples was taken from a paper by W. F. Roeser, *Jour. Applied Phys.*, Vol. 11, p. 388, 1940. Also published in "Temperature—Its Measurement and Control in Science and Industry," Reinhold Publishing Corporation, New York.

at 30°C . We may add the observed e.m.f. E_x to E_{30} (the e.m.f. given by the thermocouple when one junction is at 0° and the other at 30°C .) and obtain from the curve the true temperature t_A of the measuring junction. Certain types of instruments which are used with thermocouples in the manner shown in Fig. 5 are such that they can be adjusted manually for changes in the reference junction temperature and the e.m.f. E_A read directly on the instrument.

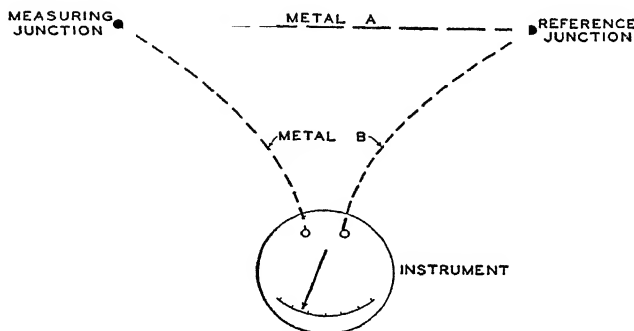


FIG. 5.—Simple thermoelectric thermometer.

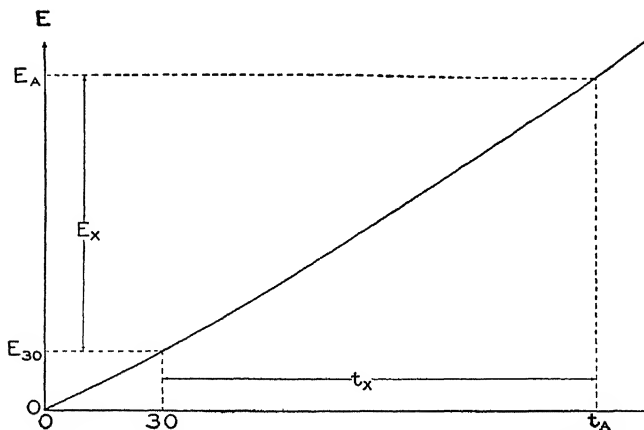


FIG. 6.—Corrections for variations in the temperature of the reference junction.

Inasmuch as the curves giving the relation between e.m.f. and temperature are not, in general, straight lines, equal increments of temperature do not correspond to equal increments of e.m.f.

In many cases, the thermocouple is connected to the instrument by means of copper leads as shown in Fig. 7.

If the junctions C and C' are maintained at the same temperature, which is usually the case, the circuit shown in Fig. 7 is equivalent to that shown in Fig. 5. If the junctions C and C' are not maintained at the same temperature, the resultant thermal

e.m.f. in the circuit will depend not only on the thermocouple materials and the temperature of the measuring junction, but also on the temperatures of these junctions and the thermoelectric properties of copper against each of the individual wires. Such a condition should be, and usually is, avoided.

Circuits such as shown in Figs. 5 and 7 are used extensively in laboratory work where it is usually convenient to maintain the reference junctions either at 0°C . by placing them in glass tubes inserted in a thermos bottle filled with cracked or shaved ice and distilled water, or at some other conveniently controlled temperature.

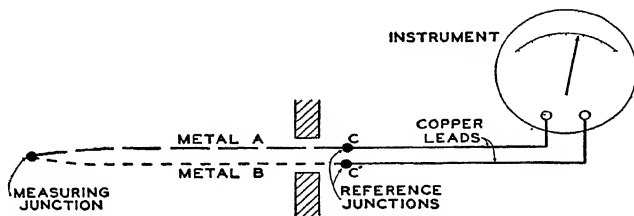


FIG. 7.—Thermoelectric thermometer with copper leads from thermocouple to instrument.

In most commercial installations where it is not convenient to maintain the reference junctions at some constant temperature, each thermocouple wire is connected to the instrument with a lead of essentially the same chemical composition and thermoelectric characteristics as the thermocouple wire, in the manner shown in Fig. 8. This is equivalent to using a thermocouple with the reference junctions at the instrument terminals. Leads that have the same thermoelectric characteristics as the thermocouple wires are called "extension leads." In most installations of this nature, the instrument is equipped with an automatic reference junction compensator which automatically changes the indication of the instrument to compensate

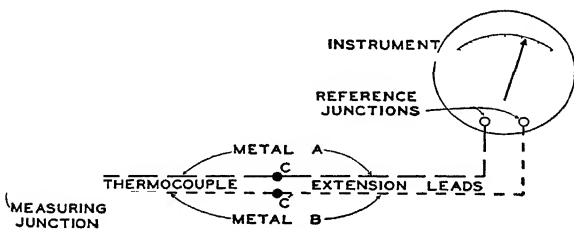


FIG. 8.—Thermoelectric thermometer with extension leads.

for changes in the temperature of the reference junctions, thus eliminating the necessity of measuring or controlling the reference junction temperature. Such automatic devices are usually part of the instrument, and in such cases the reference junctions should be located in or at the instrument or at some point that is at the same temperature as the instrument.

In some cases where expensive thermocouple wires are used, extension lead wires of less expensive materials are available which give practically the same temperature-e.m.f. relation as the thermocouple over a limited temperature range, usually 0 to 100°C . Although the combined leads give practically the same temperature-e.m.f. relation as the thermocouple wires, the individual lead wires are not identical thermoelectrically with the thermocouple wires to which they are attached, and therefore

the two junctions where the leads are attached to the thermocouples (*C* and *C'* in Fig. 8) should be kept at the same temperature. This is not necessary in the case of thermocouples where each lead and each thermocouple wire to which it is attached are of the same material.

Types of Thermocouples.—Although a thermal e.m.f. is developed when the junctions of any two dissimilar metals are maintained at different temperatures, only certain combinations of metals have been found suitable for use in the measurement of temperatures.

The combinations of metals and alloys extensively used as thermocouples for the measurement of temperatures in this country are listed in Table 5, together with the temperature ranges in which they are generally used and the maximum temperature at which they can be used for short periods. The period of usefulness of a thermocouple depends upon such factors as the temperature, diameter of wires, accuracy required, and conditions under which it is used.

TABLE 5.—TYPES OF THERMOCOUPLES AND TEMPERATURE RANGES IN WHICH THEY ARE USED

Type of thermocouple	Usual temperature range		Maximum temperatures	
	°C.	°F.	°C.	°F.
Platinum to platinum-rhodium.....	0-1450	0-2650	1700	3100
Chromel-P to alumel.....	-200-1200	-300-2200	1350	2450
Iron to constantan.....	-200- 750	-300-1400	1000	1800
Copper to constantan.....	-200- 350	-300- 650	600	1100

There are two types of platinum to platinum-rhodium thermocouples used in this country, the platinum to 90 platinum-10 rhodium and the platinum to 87 platinum-13 rhodium. These thermocouples develop, at high temperatures, 10 to 14 microvolts per °C. as compared with 40 to 55 for the other thermocouples listed in Table 5. The platinum to platinum-rhodium thermocouples at temperatures from about 400 to 1600°C. being more stable than any other combination of metals are used (1) for defining the International Temperature Scale from 660°C. to the freezing point of gold, 1063°C. (only the platinum to 90 platinum-10 rhodium thermocouple is used for this purpose); (2) for very accurate temperature measurements from 400 to 1500°C.; and (3) for temperature measurements where the lower melting point materials cannot be used. They are not suitable for temperature measurements below 0°C. because the thermoelectric power (dE/dT) is only about 5 microvolts per °C. at 0°C. and decreases to zero at about -138°C.

The nominal composition of the chromel-P alloy is 90 per cent nickel and 10 per cent chromium. Alumel contains approximately 95 per cent nickel, with aluminum, silicon, and manganese making up the other 5 per cent. Chromel-P-alumel thermocouples, being more resistant to oxidation than the other base-metal thermocouples listed in Table 5, are generally more satisfactory than the other base-metal thermocouples for temperature measurements from about 650 to 1200°C. (1200 to 2200°F.). The life of a No. 8 gauge (0.128 in.) Chromel-P-alumel thermocouple is about 1000 hr. in air at about 1150°C. (2100°F.).

Constantan was originally the name applied to copper-nickel alloys with a very small temperature coefficient of resistance but it has now become a general name that

covers a group of alloys containing 60 to 45 per cent copper and 40 to 55 per cent nickel (with or without small percentages of manganese, iron, and carbon), because all the alloys in this range of composition have a very small temperature coefficient of resistance. Constantan thus includes the alloys made in this country under such trade names as Advance (Ideal), Copel, Copnic, Cupron, etc., most of which contain approximately 55 per cent copper and 45 per cent nickel.

Iron-constantan thermocouples give a slightly higher e.m.f. than the other base-metal thermocouples in Table 5. They are extensively used at temperatures below about 760°C. (1400°F.). The life of a No. 8 gauge iron-constantan thermocouple is about 1000 hr. in air at about 760°C. (1400°F.).

Copper-constantan thermocouples are generally used for accurate temperature measurements below about 350°C. (660°F.). They are not suitable for much higher temperatures in air because of the oxidation of the copper.

Combinations of metals other than those listed in Table 5 are sometimes used for special purposes. For example, at temperatures above -200°C. (-300°F.) chromel-P-constantan gives a thermal e.m.f. per degree somewhat greater than that of any of the thermocouples listed in Table 5 and is sometimes used when the greater e.m.f. is required. Graphite to silicon carbide has been recommended [3] for temperatures up to 1800°C. (3300°F.) and for certain applications in steel plants.

Reproducibility of Thermocouples.—One of the first requirements of thermoelectric pyrometers for general industrial use is that the scales of the instruments shall be graduated to read temperature directly. Although the indications of the measuring instruments used with thermocouples depend on the resultant e.m.f. developed in the circuit, the scale of the instrument can be graduated in degrees of temperature by incorporating a definite temperature-e.m.f. relation into the graduation of the scale. The temperature can then be read directly if the temperature-e.m.f. relation of the thermocouple is identical with that incorporated in the scale of the instrument.

All the thermocouples that have the same nominal composition do not give identical relations between e.m.f. and temperature. As a matter of fact, in most cases, two samples of metal that are identical as far as can be determined by chemical methods are not identical thermoelectrically. This is due, in part, to the fact that the thermoelectric properties of a metal depend to some extent on the physical condition of the metal.

It is not practicable to calibrate the scale of an instrument in accordance with the temperature-e.m.f. relation of a particular thermocouple and to change the scale each time the thermocouple is replaced. Consequently, the scales of such instruments are calibrated in accordance with a particular temperature-e.m.f. relation which is considered representative of the type of thermocouple, and new thermocouples are purchased or selected to approximate the particular temperature-e.m.f. relation.

If the temperature-e.m.f. relations of various thermocouples of the same type are not very nearly the same, corrections must be applied to the readings of the indicator, and the corrections will be different for each thermocouple. When several thermocouples are operated with one indicator, and when thermocouples are frequently renewed, the application of these corrections becomes very troublesome. For extreme accuracy, it is always necessary to apply such corrections, but for most industrial processes thermocouples can be manufactured or selected with temperature-e.m.f. relations that are so nearly the same that the corrections become negligibly small.

The accuracy with which the various types of thermocouple materials can be selected and matched to give a particular temperature-e.m.f. relation depends on the materials and the degree to which the temperature-e.m.f. relation is characteristic of the materials available. The differences in the temperature-e.m.f. relations of new

platinum to platinum-rhodium thermocouples available in this country rarely exceed 4 to 5°C. at temperatures up to 1200°C. Consequently, there is no difficulty in selecting a relation between e.m.f. and temperature that is adequately characteristic of these thermocouples. The temperature-e.m.f. relations used in this country for platinum to platinum-rhodium thermocouples are such that new thermocouples which yield these relations within 2 or 3°C. up to 1200°C. are readily available.

The differences in the temperature-e.m.f. relations of base-metal thermocouples of any one type are so large that the selection of a temperature-e.m.f. relation that might be considered characteristic of the type of thermocouple is difficult and more or less arbitrary. The relations generally used for some of these thermocouples by some manufacturers have been changed from time to time because of differences introduced in the thermoelectric properties of the materials by variations in raw materials and methods of manufacture. However, the relations in use at the present time are such that materials can generally be selected and matched to yield the adopted relations with an accuracy of about $\pm 3^\circ\text{C}.$ up to 400°C. and to $\pm \frac{3}{4}$ per cent at higher temperatures. In special cases, materials may be selected to yield the adopted relations within 2 or 3°C. for limited temperature ranges.

Temperature-e.m.f. Relations.—Corresponding values of temperature and e.m.f. that are considered characteristic of the various types of thermocouples are given in Table 6. More detailed tables will be found in the references given. As far as we know, the corresponding values given for platinum to 90 platinum-10 rhodium, platinum to 87 platinum-13 rhodium, and chromel-P-alumel thermocouples are more or less standard and are the only ones used in this country for these types of thermocouples. The temperature-e.m.f. relations of chromel-P-alumel thermocouples have been fairly well controlled because these materials are manufactured primarily with controlled thermoelectric properties for thermocouples by only one company.

The values for iron-constantan headed *A* were determined at the National Bureau of Standards a few years ago as being characteristic of the iron and constantan generally available at that time. The values for iron-constantan headed *B* have been used by certain pyrometer manufacturers for a number of years and presumably are characteristic of the materials available at the time the corresponding values were determined. Owing to the differences in the thermoelectric properties of different lots of iron and constantan, materials must be selected and properly matched in order to obtain a thermocouple that will approximate a selected temperature-e.m.f. relation for this type of thermocouple.

Until recent years, copper-constantan thermocouples were used primarily for accurate measurements at temperatures below about 350°C., and in such cases it is customary to calibrate each thermocouple or lot of wire and use an instrument calibrated in millivolts. However, in recent years, there has been an increasing demand for direct-reading instruments for use with copper-constantan thermocouples. There are large differences in the temperature-e.m.f. relations of copper-constantan thermocouples, and consequently the materials must be selected in order to obtain thermocouples that will yield any specified temperature-e.m.f. relation. No difficulty has been encountered in obtaining thermocouples that will give very closely the temperature-e.m.f. relation given in Table 6.

Instruments.—Instruments used to measure the e.m.f. developed by thermocouples or to indicate the temperature of the measuring junction of a thermocouple may be divided into two general classes: (1) those operating on the galvanometric principle, such as ordinary millivoltmeters; and (2) those operating upon the potentiometric principle. At one time, there was considerable interest in a class of instruments that operate upon a combination of the galvanometric and potentiometric principles, but at the present time there appears to be no place or demand for such instruments.

TABLE 6.—CORRESPONDING VALUES OF TEMPERATURE AND E.M.F. FOR VARIOUS TYPES OF THERMOCOUPLES

Temp., °C.	90 Pt-10 Rh to Pt, [4] mv.	87 Pt-13 Rh to Pt, [4] mv.	Chromel-P to alumel, [5] mv.	Fe to constantan		Cu to constantan, [8] mv.
				A, [6] mv.	B, [7] mv.	
-200	-5.75	-8.27	-5.539
-100	-3.49	-4.82	-3.349
0	0.000	0.000	0.00	0.00	0.00	0.000
+100	0.643	0.646	+4.10	+5.40	+5.28	+4.276
200	1.436	1.464	8.13	10.99	10.78	9.285
300	2.315	2.394	12.21	16.56	16.30	14.859
400	3.250	3.398	16.39	22.07	21.82	20.865
500	4.219	4.454	20.64	27.58	27.39	
600	5.222	5.561	24.90	33.27	33.16	
700	6.260	6.720	29.14	39.30	39.19	
800	7.330	7.927	33.31	45.72	45.48	
900	8.434	9.177	37.36	52.29	51.82	
1000	9.569	10.470	41.31	58.22	58.16	
1100	10.736	11.811	45.14	64.50	
1200	11.924	13.181	48.85			
1300	13.120	14.562	52.41			
1400	14.312	15.940	55.81			
1500	15.498	17.316				
1600	16.674	18.679				
1700	17.841	20.032				

Millivoltmeters.—A millivoltmeter consists of a coil of wire suspended between the poles of a permanent magnet so that the coil is free to move. A pointer is attached to the coil and moves over a scale graduated in millivolts or in degrees. Leads from the thermocouple are connected to the terminals of the coil, usually through a series resistance, and the e.m.f. generated in the thermocouple circuit sends a current through the coil, causing it to deflect in the magnetic field. The magnitude of the deflection depends on the current through the coil which, in turn, depends on the e.m.f. generated by the thermocouple and the resistance of the circuit.

The current I in the circuit is given by

$$I = \frac{E}{R_0 + R_x}$$

where E is the resultant e.m.f. in the circuit, R_0 the resistance of the millivoltmeter (including series resistance, if any), and R_x the resistance of the thermocouple and leads. The potential difference E_0 across the terminals of the instrument is given by

$$E_0 = \frac{R_0}{R_0 + R_x} E$$

Millivoltmeters are ordinarily calibrated to indicate E correctly when connected to a thermocouple and leads of combined resistance R_x . Any change then in either R_0

or R_x causes a change in the indications of the instrument. Inasmuch as instruments are frequently used with more than one thermocouple and inasmuch as R_x varies with the temperature and the amount of the wire heated, it is desirable to make R_o large compared with R_x or at least to variations in R_x . However, this cannot be accomplished in most instances because the design of a millivoltmeter for any particular service is a compromise between sensitivity and ruggedness, which to a large extent determines the resistance. Millivoltmeters with a resistance of much over 600 ohms are extremely delicate and cannot be used in many locations. In order to obtain the ruggedness required, it is necessary in some instances to make R_o as low as 12 or 15 ohms. This means that variations in the resistance of the circuit have an appreciable effect on the indications of the instrument. Owing to these inherent sources of error, millivoltmeters are not extensively used at the present time.

Potentiometers.—The fundamental principle of the potentiometer may be seen by referring to Fig. 9. The current from a service cell, which may be a dry cell or a storage cell, passes through the main circuit composed of a series of fixed resistors and

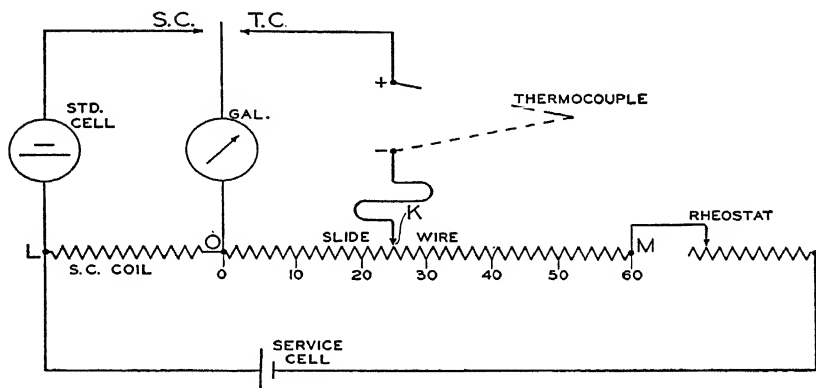


FIG. 9.—Diagram of a potentiometer circuit.

can be adjusted by a variable rheostat. The relative values of the fixed resistors and the magnitude of the current determine the range of the instrument. By means of a switch, either the standard cell or the thermocouple can be connected to the main circuit and its voltage balanced, through a galvanometer, against an equal voltage drop developed in a portion of the main circuit. The potential drop across the various fixed resistors is made to correspond to definite and reproducible values of voltage by adjusting the current to some standard value.

To standardize the battery current, the switch *SC* is closed and the current adjusted until the potential drop across the resistor *LO* is equal to the e.m.f. of the standard cell. The e.m.f. of the thermocouple can then be determined by closing the *TC* switch and moving the contact *K* until the potential drop in the resistors in the main circuit from *O* to *K* is equal to the e.m.f. of the thermocouple. The potential drop from *O* to *K* is read on some suitable scale. The resistors included between *O* and *M* may be made up of fixed coils, a slide wire, or a combination of the two.

The galvanometer has no measuring function but serves solely to indicate zero current in its branch of the circuit. As there is no current through the galvanometer and thermocouple when a balance is obtained, the reading of the potentiometer is

independent of the resistance of the thermocouple circuit. However, large changes in this resistance affect the precision of balancing, because as the resistance is increased the sensitivity of response of the galvanometer is decreased and the greater is the change in e.m.f. required to produce a perceptible deflection of the galvanometer. It is therefore advisable to keep the resistance of the thermocouple and leads within reasonable limits as determined by the characteristics of the galvanometer.

There are several important advantages in the potentiometer method. The scale is easily made very open, thus permitting precise readings if a suitable galvanometer is used. The accuracy of the potentiometer is in no way dependent upon the constancy of magnets, springs, or jewel bearings, nor upon the level of the instrument. Insofar as temperature measurements with thermocouples are concerned, the greatest advantages are the accuracy and the complete elimination of any error due to ordinary changes in the resistance of the thermocouple and leads. The objections to the potentiometer are its higher initial cost and the fact that, except for automatic instruments, a manual adjustment must be made to obtain a reading. In the recording and controlling potentiometer equipment, however, all the various manipulations may be performed automatically even to the standardizing of the battery current.

Potentiometers are used for practically all laboratory work where the highest accuracy is required. At the present time, the large part of the instruments in industrial use operate upon the potentiometric principle.

The scales of millivoltmeters or potentiometers may be calibrated either in millivolts or in degrees of temperature for a particular type of thermocouple. Either type of instrument can be made automatically recording and can be equipped with devices for automatically controlling the temperature of a process or furnace.

Inasmuch as the e.m.f. developed by a thermocouple depends upon the temperature of the reference junctions as well as upon the temperature of the measuring junction, corrections must be made for changes in the reference-junction temperature unless they are automatically taken care of by the instrument. If properly calibrated, an ordinary millivoltmeter will indicate the temperature of the measuring junction, if the pointer of the indicator is set to read the reference-junction temperature on open circuit. If the reference-junction temperature is nearly constant, this adjustment may be made conveniently by hand. However, if this junction is subject to frequent temperature changes, it is advisable to locate the reference junctions at the instrument and use an instrument that automatically changes the indication to compensate for changes in the reference-junction temperature. In the case of millivoltmeters, this is accomplished automatically by attaching a properly adjusted bimetallic spring to the control spring of the moving coil. If an instrument is equipped with an automatic reference-junction compensator for one type of thermocouple, it should not be used with other types of thermocouples, nor should it be used in any case unless the reference junctions are located at the instrument.

Potentiometers for industrial use are usually provided with an automatic reference-junction compensator or in special cases a hand-operated compensator. A hand-operated compensator is an adjustable resistor, having a scale graduated in millivolts or in degrees, which the operator sets to correspond with the observed reference-junction temperature. This operation is equivalent to moving the fixed end of the thermocouple circuit (point O, Fig. 9) to a point corresponding with the reference-junction temperature. For use with any one type of thermocouple, the compensator may include a nickel coil which varies in resistance as the temperature changes and thus compensates automatically for changes in the reference-junction temperature. An instrument so equipped is usually calibrated directly in degrees of temperature. Although it is usually at the instrument, an automatic compensator can be located wherever desired. Lead wires from the thermocouple to the compensator location

in any case must have the same thermoelectric characteristics as the thermocouple wires.

Protection Tubes.—One of the reasons why the materials listed in Table 5 have come into common use for thermocouples is that they are reasonably stable thermoelectrically when heated in a clean oxidizing atmosphere. The standards of performance that are generally accepted for the various combinations of thermocouple materials are based on their performance in air. Although it has not been shown that a reducing atmosphere, in itself, necessarily contaminates or alters the thermoelectric properties of thermocouple materials, it is nevertheless generally observed that exposure of thermocouples to such atmospheres is accompanied by contamination or changes in the chemical composition which seriously alter the thermoelectric properties. In order to obtain the best performance of thermocouples, it appears necessary to maintain them in an atmosphere having essentially the same composition as air. Consequently, the selection of a proper protection tube, which will protect the thermocouple from vapors, fumes, or furnace gases, is sometimes as important as the selection of the thermocouple materials.

Changes in thermoelectric characteristics result from causes such as the following;

1. Metals (solid, liquid, or vapor) coming into contact with the thermocouple materials and altering their chemical composition.

2. Furnace gases and fumes coming into contact with the thermocouple materials. Sulphur and sulphur compounds are particularly deleterious.

3. Materials normally stable in an oxidizing atmosphere coming into contact with the thermocouples in a reducing atmosphere. One common cause of contamination, which is serious in the case of rare-metal thermocouples, is the reduction of silica (usually present in insulating and ceramic protection tubes) to silicon which readily combines with the thermocouple materials.

4. Preferential oxidation and reduction of base-metal alloys exposed alternately to oxidizing and reducing atmospheres. This results in a gradual change in chemical composition, because all the elements that comprise an alloy are not oxidized and reduced at the same rates under all conditions.

Many types of protection tubes are required in order to meet all needs. In many cases, the conditions under which thermocouples are used are such that two tubes are required to provide the desired protection. For instance, a primary tube of porcelain or fused silica may be placed inside a secondary tube of metal, silicon carbide, or fire clay. The primary tube of low volatility is intended to provide imperviousness to gases at high temperatures, and the secondary tube to provide resistance to thermal and mechanical shock and to corrosion.

In thus providing protection for the thermocouple, however, one should not lose sight of the fact that a thermocouple can perform its function only when the conditions of heat transfer are such that the measuring junction approaches the temperature to be measured within the accuracy desired. When a tube of large cross section or more than one tube is used, particularly if the tubes have a high thermal conductance, it should be carefully considered whether the depth of immersion is sufficient to ensure that the temperature of the thermocouple junction is reasonably close to the temperature to be measured. Short thick-walled tubes may cool the junction so much that the indications are comparatively worthless.

Platinum to platinum-rhodium thermocouples are particularly susceptible to contamination and should be protected by ceramic tubes which are impervious to gases and vapors at all operating temperatures. Metal protection tubes usually provide sufficient protection for base-metal thermocouples. The oxide coatings on the thermocouple wires are fairly effective in protecting the wires from contamination by metallic vapors. Metal tubes that provide sufficient protection in an oxidizing

atmosphere may be entirely unsatisfactory if large amounts of furnace gases are present. In some installations, it has been found advisable to ventilate the interior of the protection tube with a slow stream of air in order to minimize the deleterious effects of furnace gases.

The primary ceramic tubes, which meet most requirements of stability and imperviousness to gases, are highly refractory porcelain, sometimes called Sillimanite or Mullite for temperatures up to about 1550°C. (2800°F.); fused silica for temperatures up to about 1050°C. (1900°F.) in an oxidizing atmosphere; and Pyrex glass for temperatures up to about 600°C. (1100°F.).

The secondary, or metal, tube most suitable for a particular application depends to a large extent on the type of corrosion encountered. Nickel-chromium-iron tubes are particularly useful in oxidizing atmospheres, chromium-iron tubes in atmospheres containing sulphur, and nickel or iron tubes in hot caustic and molten-metal baths. The temperature limits given in Table 7 for the various types of tubes are those which will, in general, result in a reasonably long life. The tubes may be used at higher temperatures than those given, but higher operating temperatures will result in a shorter life.

Fire clay, silicon carbide, and graphite meet certain requirements of secondary tubes at temperatures above the useful limits of metal tubes. Numerous other types of tubes have been developed for specific purposes. Recommendations regarding tubes for any particular purpose may be obtained from pyrometer manufacturers.

Measurement of Temperatures.—The measurement of the temperature of any particular object or space is in many cases a problem in itself, even after a suitable thermocouple has been selected and calibrated.

TABLE 7.—RECOMMENDED MAXIMUM OPERATING TEMPERATURE OF METAL PROTECTION TUBES

Type of tube	Recommended maximum temperature	
	°C.	°F.
Seamless steel.....	550	1000
Wrought iron.....	700	1300
Cast iron.....	700	1300
Calorized wrought iron.....	800	1500
14 per cent chromium iron..	800	1500
28 per cent chromium iron..	1100	2000
18 (Cr)-8 (Ni)-stainless steel	850	1600
32 Ni-20 Cr-48 Fe.....	1100	2000
62 Ni-13 Cr-25 Fe.....	1150	2100
Nickel.....	1100	2000

The temperature indicated by a thermocouple is that of its measuring junction, but usually this is of no interest in itself. The accuracy obtained in measuring the temperature of any object or space usually depends on how closely the junction of the thermocouple can be brought to the same temperature as that of the object or space, or to some temperature which is definitely related to that of the object or space.

If under steady conditions there is a net exchange of heat between the thermocouple junction and an object, then there is a difference in temperature between the

two. The magnitude of this difference in temperature depends on the rate of heat transfer and the thermal resistance between the junction and the object. The idea then is to bring the thermocouple junction into as good thermal contact as possible with the object and to insulate the junction as well as possible against the transfer of heat to or from other objects or spaces. Greater precautions are obviously necessary in accurate measurements than in rough ones.

As an illustration, suppose we desire to measure the temperature of a metal plate that is heated from within by some means. The bare thermocouple junction is brought into contact with the metal plate. The junction will receive some heat from the plate by thermal conduction and probably a smaller amount by radiation and convection. The junction will lose heat by conduction along the thermocouple wires, and by convection, conduction, and radiation to the surroundings. Obviously the junction will be at a lower temperature than the plate. However, this difference in temperature can be reduced by the following methods:

1. By improving the thermal contact. (a) By flattening the junction to obtain a larger area of contact, or better still (b) by soldering, brazing, or welding the junction to the plate.

2. By reducing the heat loss from the junction. (a) By keeping the wires close to the plate for some distance so as to reduce the temperature gradient in the wires near the junction, and/or (b) by raising the temperature of the space immediately surrounding the junction (1) by insulating the junction from that space or (2) by utilizing an auxiliary source of heat as is done in the compensated contact thermometer.

The thermocouple junction may be at either a higher or a lower temperature than the object, depending on the direction of the net heat transfer. The use of thermocouple protection tubes usually makes it more difficult to bring the thermocouple junction to nearly the same temperature as that of an object, because of the additional thermal resistance introduced between the junction and object and the additional transfer of heat along the protection tubes. For example, the junction of a thermocouple may be brought within a few hundredths of a degree of the temperature of a liquid by immersing the bare wires in the liquid for a distance of 5 to 10 diameters of the wire, whereas if a protection tube is used with the thermocouple it will be necessary to immerse the junction 5 to 10 diameters of the protection tube to obtain the same degree of accuracy. In most applications, the best that can be done is to bring the thermocouple junction and object as close together as possible and immerse the thermocouple as far as practicable in the heated medium.

In the measurement of certain temperatures, particularly those of small objects and materials of low thermal conductivity, consideration must be given to the possibility that the temperature to be measured may be altered by the introduction of the temperature-measuring device.

In the measurement of temperatures varying with time, heat capacity of thermocouples and protecting tubes as well as thermal contacts are involved. If the temperature is rising, the temperature of the junction itself will at any instant be lower than that of the surroundings. The reverse will be true if the temperature is falling. This is referred to as "thermometric lag." By appropriate auxiliary measurements, however, it is usually possible to determine the lag [9] under the particular conditions of use and apply corrections to the thermocouple readings.

Installations.—The cost of pyrometer equipment is great enough to warrant installing it, not only so that the desired temperature can be measured with the required accuracy, but also in such a manner as to protect the equipment, as far as practicable, from changes and deterioration. The installation of simple pyrometer equipment for more or less temporary service in an air-conditioned laboratory, far removed from the vibrations set up by heavy equipment such as trip hammers and

rolling mills is a comparatively simple matter. However, in the large installations required in many industrial plants, every precaution should be taken to protect the thermocouple, wiring, and instruments from the various deleterious conditions that may seriously affect their accuracy and life.

General Precautions.—The installation of extensive thermocouple equipment requires the services of competent electricians. Just as much attention, if not more, should be given to the wiring, switches, switchboards, etc., as is given in the case of ordinary power installations. Proper fixtures should be used to mount the thermocouples in the furnaces. Lead wires should have a weatherproof covering and should be run in a metal conduit except possibly for a short length of flexible cable at the ends of the conduit. The conduit should be grounded to prevent leakage from power installations or lighting circuits. All joints in the lead wires should be soldered and taped. When indicators or recorders of low resistance are employed, it is of the greatest importance to have a well-constructed electrical installation to ensure a constant line resistance. Since instruments of low resistance are usually calibrated for a low line resistance of definite value, special attention must be given to contact resistances at switches. Frequently switches rated at 100 amp. are required, although the actual thermoelectric current is only a few milliamperes. If the indicator is of high resistance, or operates on the potentiometric principle, the factor of very low line resistance is not of great importance, but the wiring should be well installed, to avoid large changes in the resistance. Stationary indicating and recording instruments usually should be mounted upon switchboards, with suitable selective or commutating switches when several thermocouples are to be used with one indicator. When the junction between the thermocouple and leads is exposed to severe conditions, rain, etc., as in outside kilns, a weatherproof terminal head should be used. This consists of an outside casing that fits over both binding posts. The cover may be tapped for conduit wiring or provided with a packing gland or stuffing box if a length of flexible cable is used between the thermocouple and the conduit.

The indicator or recorder should be conveniently located. If the instrument is desired especially for the use of the operator of a furnace, it should be placed where it is readily visible. It should be mounted where vibration and excessive dirt and dust will not injure delicate parts of the mechanism. In almost all industrial installations, an outside protecting case is required to prevent dust from filtering through the case of the indicator. Special devices are employed to damp out vibration when this is serious, as in the neighborhood of a trip hammer or rolling mill. Frequently the instruments are suspended on "vibration-proof mountings."

Location of Thermocouples.—The proper location of the thermocouple in a furnace depends on the particular process and the use to which the furnace is put. The primary consideration is to locate the measuring junction so that it acquires the temperature it is desired to measure. This requires taking into consideration the thermal lag and heat transfer to and from the thermocouple junction. Both of these factors are materially affected by protection tubes. A secondary consideration, however, is to locate the thermocouple where the lead wires may be conveniently located. The space between the protecting tube and the furnace wall should be plugged with refractory cement, so that there can be no leakage of hot gases through the hole onto the terminals of the thermocouple, nor can cold air be drawn in, thus cooling the measuring junction of the thermocouple.

Common Return.—The use of a common return wire for a multiple installation is in general unsatisfactory. When grounds or short circuits occur at any point between the measuring junctions and the instrument—*e.g.*, between the common return and the other lead wire of one thermocouple—all the thermocouples on the common return have, in addition to their own *e.m.f.*, an impressed potential drop due

to the current flowing in the shorted thermocouple. This may cause a large error in the reading of every thermocouple in the system. With the common return, leakage from a power installation affects the readings of every thermocouple connected to the return, and a leakage through a high resistance may alter the readings of every thermocouple by the same amount so that the presence of such leaks is not always readily detected. It is also possible by leakage from different thermocouples to the ground to obtain very erratic and erroneous readings when the common return is employed. The insulation resistance of materials becomes very low at high temperatures, making it difficult to insulate the various thermocouple wires from one another and from the furnace parts, even when individual returns are used. Base-metal thermocouples are frequently constructed with the measuring junction welded to the end of the metal protecting tube in order to reduce thermal lag. Even when the thermocouple is not welded at this point, the measuring junction usually touches the protecting tube and is in good electrical contact with it. If a common return is used when the thermocouples are mounted in this way, any electrical connection or leakage between the metal protecting tubes (through the furnace structure, etc.) will produce

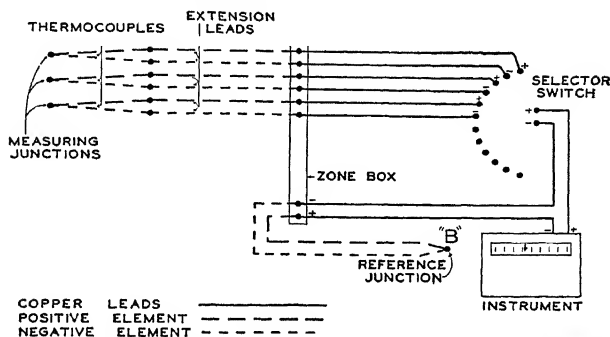


FIG. 10.—Connections between several thermocouples and a distant instrument.

serious errors. However, if individual returns are used, the metal protecting tubes need not be insulated from one another. The troubles ordinarily encountered with common return installations are sufficient to warrant the extra cost of the wire required to install individual thermocouple circuits.

Use of a Zone Box.—Extension lead wires are rather costly and should not be employed extravagantly. Also the use of long lengths of extension leads increases the line resistance, since the resistivity of the lead materials is much higher than that of copper.

For an installation in which several thermocouples are connected through a selector switch to an instrument located at some distance from a furnace, a wiring scheme similar to that shown in Fig. 10 may be very useful in saving extension lead wire and, consequently, in reducing the cost and resistance of the line. The common reference junction *B* for each thermocouple can be located at any conveniently controlled point or at the instrument if the latter is equipped with an automatic reference-junction compensator by running only one pair of extension leads from the zone box to the controlled point or the instrument. A zone box is a box or zone of uniform temperature which need not be measured or maintained constant. It should be pointed out that, although a common reference junction is used for all the thermo-

couples, the system is not subject to any of the objections that have been raised against the use of a common return.

Thermocouples Purposely Insufficiently Immersed.—In many processes, a furnace is operated at such a high temperature that a thermocouple or protecting tube cannot withstand the severe conditions inside the furnace. In this case, the thermocouple may be advantageously inserted only part way through the furnace wall, or flush with the inner wall of the furnace. The temperatures indicated by thermocouples installed in this manner are always lower than those of the furnace interior, but they may bear a fairly definite relationship to the temperature of the furnace, and hence the method may be satisfactory for temperature control and reproduction of furnace conditions from day to day.

OPTICAL AND RADIATION PYROMETRY

General Theory.—The temperature of a material may be obtained from a measurement of the intensity of the radiant energy it emits. This measurement may refer to the radiation of all wave lengths emitted by the material, or if the material is glowing, the measurement may refer to the visible light emitted, or to the radiation in a very restricted portion of the spectrum. However, in general, the intensity of radiation depends not alone on the temperature of the source, but also on the particular material constituting the source. Thus glowing carbon appears to the eye about three times as bright as glowing platinum when both are at the same temperature. This is technically expressed by the statement that the emissive power or emissivity of carbon is about three times that of platinum. A material having the highest theoretically possible emissivity is known as a "black body." In general, it is customary to assign a numerical value of 1 to the emissivity of a black body. Hence, all other materials have an emissivity less than 1. A black body is experimentally realized by uniformly heating a hollow enclosure and observing the radiation coming from a small opening in the wall. The intensity of radiation emitted from this opening depends only on the temperature of the walls. It does not depend on the material of which the walls are constructed. If E is the emissivity of any nontransparent material and R its reflection coefficient, it can be shown that $E + R = 1$. If a material having an emissivity of, say, 0.40, and hence a reflection coefficient of 0.60, is placed inside a black body, it becomes indistinguishable from its surroundings. The total intensity of radiation leaving the material is the same as that emitted by the black body. Thus, while the material actually emits only 40 per cent of the intensity of a black body at the same temperature, under the above conditions 60 per cent of the radiation falling upon it from the walls of the enclosure is reflected, with the net result that the object appears of the same intensity as its surroundings.

However, if the material is removed from the black body and placed in the open air, the reflected intensity is no longer present and the object appears only 40 per cent as bright as a black body at the same temperature. Optical and radiation pyrometers are usually calibrated to read correctly when sighted upon a black body. Fortunately, many technical processes are carried out under black-body conditions. Muffle furnaces, many annealing furnaces, etc., are sufficient approximations to "black bodies" to give practically correct temperature readings with the optical or radiation pyrometer. Some materials in the open are nearly "black"—*e.g.*, the oxide formed on iron and steel ingots, rails, etc.

In general, however, corrections must be applied to the pyrometer readings to obtain the correct temperatures of materials in the open. These corrections are very large in the case of clean molten metals. The presence of an oxide film on the molten metal surface greatly reduces the corrections. For temperature control, it is not always necessary to correct the observed readings. As far as the factor of

emissivity is concerned, the actual pyrometer readings, although known to be too low, will always be too low by the same amount from time to time for the same observed temperature and, hence, will furnish as a general rule as good information for temperature control and uniformity as could the true temperatures.

Optical Pyrometry.—The temperature scale of any optical pyrometer is based on Wien's law for spectral distribution of the energy radiated by a black body. This law may be stated by the following equation:

$$J_{\lambda} = c_1 \lambda^{-5} e^{\frac{-c_2}{\lambda \vartheta}} \text{ for a black body} \quad (1)$$

where λ denotes the wave length in microns, c_2 a constant = 14,320 micron degrees, ϑ the absolute temperature (degrees Kelvin) of the black body, J_{λ} the intensity at the wave length λ (*i.e.*, at a particular color such as red), and c_1 a constant, the value of which is of no consequence in pyrometry, as will be seen, since it disappears from the actual working equations. The intensity of radiation J_{λ}' of wave length λ , from a nonblack body of emissivity E_{λ} at temperature ϑ , is given by Eq. (2).

$$J_{\lambda}' = c_1 \lambda^{-5} E_{\lambda} e^{\frac{-c_2}{\lambda \vartheta}} = c_1 \lambda^{-5} e^{\frac{-c_2}{\lambda S_{\lambda}}} \text{ for a nonblack body} \quad (2)$$

In the third term of Eq. (2), S_{λ} is defined as the apparent temperature in degrees absolute of the nonblack body. This is the temperature measured by an optical pyrometer and is less than the true temperature ϑ for all materials except black bodies when it becomes equal to ϑ . From Eq. (2)

$$\frac{1}{\vartheta} - \frac{1}{S_{\lambda}} = \frac{\lambda \log E}{c_2 \log e} = \frac{\lambda \log E}{6219} \quad (3)$$

In the use of any optical pyrometer, either λ is known or has been involved in the calibration, and, knowing the emissivity E_{λ} , it is possible to obtain the true temperature ϑ from the observed temperature S_{λ} . If λ is not known, it must be estimated in order to know E_{λ} at the wave length λ .

An optical pyrometer may be defined as a pyrometer with which temperature is measured by means of the change with temperature of the light emitted by a glowing body. It consists of a photometer using monochromatic light (usually red) in which the intensity of radiation from the object whose temperature is to be measured is matched or made equal to that from a variable standard or calibrated source; or, conversely, the standard source is maintained constant, and the radiation from the object is varied to match by means of absorbing devices, such as dark glass, interposed between the object (furnace) and the standard light source. In this process of comparing the intensities of light (brightnesses) of the two sources, the term $c_1 \lambda^{-5}$ of Eq. (1) is embodied as one of the calibration constants of the instrument.

Of the different types of optical pyrometer devised and used at various times, only two will be described here. Descriptions of other types can be found in treatises on pyrometry.

Disappearing-filament Optical Pyrometer.—This type of optical pyrometer is in most general use, having the advantages of accuracy, sensitivity, and portability and that it can be sighted upon small objects, such as tools or a crack or small peephole in a furnace. Figure 11 illustrates the essential elements of the disappearing-type optical pyrometer.

The filament of a small electric lamp F (Fig. 11) is placed at the focal point of an objective L and ocular forming an ordinary telescope which superposes upon the lamp the image of the source viewed. Red glass, such as Corning "high-transmission red,"

is mounted at the ocular to produce approximately monochromatic light. In making a setting, the current through the lamp is adjusted by means of a rheostat until the tip or some definite part of the filament is of the same brightness as the source viewed. The outline or detail of this section of the filament is then indistinguishable from the surrounding field. With instruments recently developed, the temperature is read directly on the scale of a potentiometer. With instruments of the older type, the current through the lamp is read on an ammeter and the corresponding temperature is obtained from a plot or table giving corresponding values of current and temperature.

The lamps should not be operated at temperatures higher than about 1400°C. on account of the deterioration of the tungsten filament. If this temperature is not exceeded, the calibration of the lamp is good for hundreds of hours of ordinary use. For higher temperatures, absorption glasses *S* (Fig. 11) are used between the lamp and objective, or in front of the objective, to diminish the observed intensity of the source.

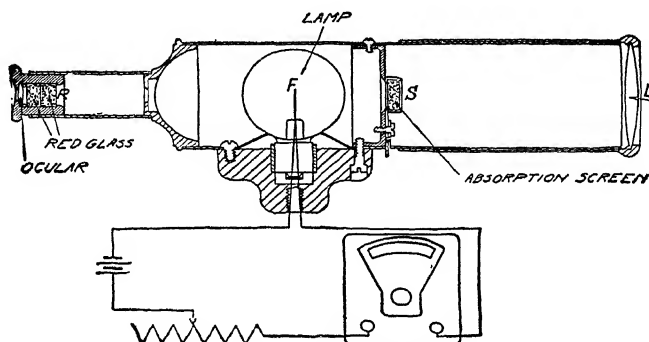


FIG. 11.—Disappearing-filament optical pyrometer.

The relation between the temperature of the source ϑ° absolute and the observed temperature ϑ_0° absolute measured with the absorption glass interposed is as follows:

$$\frac{1}{\vartheta} - \frac{1}{\vartheta_0} = A$$

where *A* is for most practical purposes a constant.

If the instrument is not direct reading, it is usually furnished with a table showing the relation between the current through the lamp and the temperature both with and without the absorption glass. If, however, this relation is not given when the absorption glass is used, it may be readily determined by measuring the constant *A* in the above formula. To do this, sight without the absorption glass on a muffle furnace or any uniformly heated furnace at 1200 to 1500°C., and observe the temperature ϑ , in degrees absolute. Then with the absorption glass in place, match the filament again and observe to what temperature ϑ_0 in degrees absolute the current through the lamp corresponds. The difference in the reciprocal of these two temperatures is the constant *A*. This determination should be repeated several times and at several temperatures of the furnace. The separate values of *A* should not differ by more than 1 per cent, and the mean value is used for computing the relation between the observed absolute temperature with the absorption glass and the true temperature of the source. In making these computations, all temperatures must be converted to degrees absolute. Practically any desired temperature range extending above 1400°C

can be obtained by employing absorption glasses of the required thickness. In this type of optical pyrometer, the current required to cover the usual temperature ranges is from about 0.32 to about 0.54 amp. Consequently, in an ammeter with a scale reading from 0 to 0.6 amp., the lower half of the scale is not used and the upper half cannot be read with sufficient accuracy for most purposes. In some types of ammeters, the zero is depressed so that if the instrument is to read 0.55 amp. at full scale the lowest mark on the scale will be about one-half this value.

The principal advantages of the recently developed potentiometer type of portable optical pyrometer, in addition to being direct reading, are (1) the lamp filament is made much smaller, thereby reducing the size and weights of batteries as well as the thermal lag and the effect of changes in the ambient temperature; and (2) the scale is made much longer than can be obtained with a deflection-type instrument, thereby permitting more accurate readings.

Figure 12 illustrates the arrangement of optical parts in the original form of another type of optical pyrometer. Light from the furnace is focused at the center of the silver-strip cube *C*. This cube produces a circular field divided through the middle. One-half of the field receives light from the furnace and the other half from the ground-glass screen *D* which is illuminated by the electric lamp *F* through the condenser

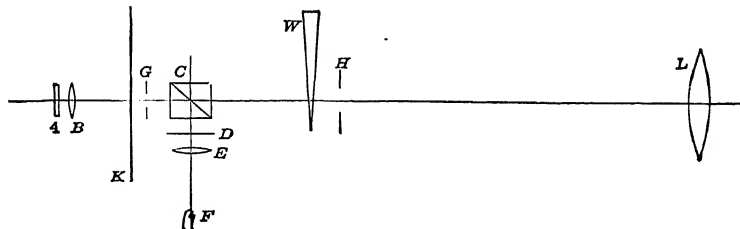


FIG. 12.—F. & F. pyrometer.

lens *E*. The ocular containing the red-glass screen *A* and lens *B* is focused on the dividing edge of this photometric field. *G* and *H* are diaphragms that limit the cone of rays employed. The two fields are matched by turning a thumbscrew that moves the black-glass wedge *W* across the path of the light from the furnace. By a system of gears, this movement is transferred to a circular scale on the dial *K* of the instrument. In the present form of the instrument, the linear wedge has been replaced by a semicircular wedge, thus eliminating the gears. Also the comparison lamp has been mounted in the line of sight, replacing the cube *C* in Fig. 12. In this arrangement, the lamp is held at a constant specified current and a match is obtained by rotating the wedge until the tip of the lamp disappears or matches the background. The instrument in its later form is, in reality, a disappearing-filament pyrometer, although, because of the effect of the wedge, the filament is usually visible except over a very short length at its tip, when a match of brightness is made. In order to increase the range of the instrument, and also the scale length, an absorption glass is used, just as in Fig. 11. The high- and low-range scales are calibrated directly in degrees of temperature, without graduations of angular degrees. The calibration in such a case is almost entirely empirical. An ammeter of suitable sensitivity is used, with a mark, or marker, showing the normal current of the lamp. Once the instrument has been correctly calibrated, a new lamp can be substituted for an old one by merely determining the new normal current.

General Use of Optical Pyrometers.—Optical pyrometers and radiation pyrometers, described later, afford the most reliable means yet developed for measuring

temperature above 1500°C. When the instrument is especially designed for precision work, it is possible to measure temperatures to about 0.2°C. at 1100°C. and about 0.5°C. at 1800°C. These figures correspond to about $\frac{1}{4}$ per cent in brightness. As an illustration, Roeser, Caldwell, and Wensel [10] at the National Bureau of Standards obtained 1773.5°C. for the freezing point of platinum in 1931; Schofield [11] in England obtained 1773.3°C. in 1934; and Hoffmann and Tingwalt [12] in Germany obtained 1773.8°C. in 1934.

The commercial form of the instrument when properly calibrated can be relied upon to about 3°C. at 1100°C. and about 5°C. at 1800°C. Temperature differences corresponding to about 1 per cent in brightness can be readily detected. This corresponds to about 1°C. at 1100°C. and about 2°C. at 1800°C. With a well-designed optical pyrometer, there is a perfect color match of the two fields at all times. Hence, contrary to the general impression, color is not matched at all, but simply brightness of uniform color. A color-blind observer will obtain the same settings as a normal observer.

Although the optical pyrometer is essential for the measurement of temperatures above 1500°C., its usefulness is by no means confined to the high-temperature range. The thermocouple cannot be adapted to many processes at low temperatures—*e.g.*, the measurement of the temperature of steel rails as they pass through the rolls, ingots, and forgings in the open, and small sources, such as a heated wire or lamp filament. The temperatures used in the above processes may be accurately measured by the optical pyrometer. The temperature of a microscopic sample of any material can be measured by a modified form of the disappearing-filament pyrometer [13]. Also in many processes a thermocouple is not so convenient as an optical pyrometer, especially when the temperature is not required often enough to warrant a permanent installation of thermocouples.

One serious objection to the optical pyrometer from the industrial point of view is the fact that it has not been made commercially for automatic recording. Since a photometric match is required for every setting, the instrument necessitates the attention of an observer. Another objection is the introduction of the human element into the readings, thus affording an opportunity for dishonest or prejudiced settings. The observer, if he is the operator of the furnace, should be taught that the instrument is for his own assistance and that it is not to be considered as a regulatory measure. Otherwise the measurements should be made by a disinterested party.

Black-body and Nonblack-body Conditions.—Optical pyrometers are usually calibrated to read correctly when sighted on a black body. Many furnaces approximate black-body conditions very satisfactorily. In a perfect black body, the details of the inside of the furnace vanish and a piece of steel, for example, which is being heated cannot be distinguished from the background. If the objects in the furnace can be distinguished but only on close observation, and if much of the detail is lost, after they have been in the furnace some time, it is not likely that the temperature measurement will be seriously in error. If in error at all, the observed temperature will be too high when the furnace walls are brighter than the material being heat treated and too low when the walls are less bright. This latter condition is possible if the heat supply is variable or if it is shut off and the furnace allowed to cool.

That a steel ingot placed in a heated furnace may appear much hotter than it really is, is a fact not always appreciated. The surface of the ingot appears hot because it reflects the bright light from the walls of the furnace. Of course, in comparison with the much greater brightness of the walls, a cold ingot appears black, but this is due to the intense contrast. If the ingot is viewed alone when the direct radiation from the furnace is screened from the eye, it also is bright. Thus when an optical pyrometer is sighted on an ingot in the furnace, part of the light reaching the

instrument comes from the side walls and is reflected by the surface of the ingot. Iron oxide reflects about the least amount of any material met with in metallurgical practice. Its emissivity is approximately 0.95; hence its reflection coefficient is 0.05. Suppose an iron ingot at room temperature were suddenly placed in a furnace at 1200°C. Although the surface of the ingot is cold, it reflects 5 per cent of the light falling upon it from the hot side walls. This 5 per cent of reflected radiation gives the ingot the appearance of an object at 950°C. The measurement by the optical pyrometer accordingly would be 950°C., although actually the ingot is at room temperature. The higher the reflecting power of the material, the greater the observed temperature under the above conditions. Thus, cold platinum would appear to be at about 1160°C., or at almost the same temperature as that of the furnace. One method for reducing the error due to reflected radiation is to view the object through a large open door on a surface parallel to the opening. If the door is large enough, the surface of the ingot thus loses a great portion of the furnace wall contributing the reflected light. The ingot reflects the image of the opening left by the door, and this opening is not radiating. It must be pointed out that the reflection from any material being heat-treated is diffuse like that from a matte surface, so that an opening in a furnace wall is not truly imaged as would be the case with a polished mirror surface. Stray reflection is, however, greatly minimized even with matte surfaces by sighting through a large door on a surface parallel to the opening. When the material has attained the temperature of the side walls, it is, of course, not desirable to open a

TABLE 8.—TRUE TEMPERATURES VS. APPARENT TEMPERATURES MEASURED BY OPTICAL PYROMETERS USING RED LIGHT ($\lambda = 0.65$ micron) WHEN SIGHTED ON THE FOLLOWING MATERIALS IN THE OPEN

Observed temperature, °C.	True temperature, °C.								
	Molten copper	Molten steel ¹	Molten cast iron		Solid iron oxide	Solid nickel oxide	Ni-chrome or chromel in air	Molten slag ²	Bright platinum
			Oxide	Clean metal					
700	700	701	705	755
800	801	802	806	867
900	902	904	907	980
950	1094	953	955	957	1037
1000	1157	1004	1007	1008	1095
1050	1220	1055	1058	1058	1152
1100	1284	1106	1110	1109	1210
1150	1349	1184	1158	1162	1160	1268
1200	1414	1236	1210	1215	1210	1326
1250	1480	1288	1267	1261	1385
1300	1546	1341	1410	1320	1312	1444
1400	1525	1447	1525	1415	1455	1563
1500	1641	1641	1565	1683
1600	1758	1670	
1700	1876	1780	
1750	1936	1830	

¹ Computed for $E_{\lambda} = 0.40$, this being the best value for ordinary steel practice.

² Computed for $E_{\lambda} = 0.65$, an average value for liquid slags.

large door, since then the opening affects the black-body conditions. The pyrometer should be sighted through a small peephole as soon as approximate temperature uniformity is obtained. A more satisfactory method for reducing the stray reflections than opening a large door is to sight into a deep wedge-shaped cavity or hole made in the metal being heat-treated. If this cavity is deep enough, very little radiation from the side walls can be reflected from it. If such a hole cannot be made conveniently, a length of iron pipe closed at one end, or a porcelain tube, may be placed on the material and so aligned that the pyrometer may be sighted through a peephole directly into the bottom of the tube.

The effect of reflected light is very noticeable in an empty coke oven. The reflection coefficient of the brick walls is comparatively high. If a perfect mirror were placed parallel to a glowing surface, the mirror would appear as hot as the surface. This effect of reflection takes place in a coke oven so that both walls appear of approximately equal brightness, even though they may differ considerably in temperature. Frequently, a patch of the wall on one side becomes coated with a layer of coke. Since the coke has a higher emissive power than brick, this patch appears much hotter. Actually, it is at about the same temperature as the less bright surrounding wall. On account of reflection, a corresponding bright patch appears on the opposite wall, although this wall may be free from coke. It is evident that the measurement of temperature of a portion of a nonuniformly heated furnace by means of an optical pyrometer is difficult unless the precautions suggested above are taken. As soon as the furnace attains temperature uniformity and equilibrium, the optical pyrometer gives the true temperature very easily and readily.

When an optical pyrometer is sighted on a glowing material in the open, it reads too low. Certain materials, important industrially, have a very high emissivity so

TABLE 9.—SPECTRAL EMISSIVITY OF MATERIALS AT $\lambda = 0.65$ MICRON; SURFACE UNOXIDIZED

Element	$\epsilon_{0.65 \mu}$		Element	$\epsilon_{0.65 \mu}$	
	Solid	Liquid		Solid	Liquid
Beryllium.....	0.61	0.61	Thorium.....	0.36	0.40
Carbon.....	0.80-0.93		Titanium.....	0.63	0.65
Chromium.....	0.34	0.39	Tungsten.....	0.43	
Cobalt.....	0.36	0.37	Uranium.....	0.54	0.34
Columbium.....	0.37	0.40	Vanadium.....	0.35	0.32
Copper.....	0.10	0.15	Yttrium.....	0.35	0.35
Erbium.....	0.55	0.38	Zirconium.....	0.32	0.30
Gold.....	0.14	0.22	Steel.....	0.35	0.37
Iridium.....	0.30		Cast iron.....	0.37	0.40
Iron.....	0.35	0.37	Constantan.....	0.35	
Manganese.....	0.59	0.59	Monel.....	0.37	
Molybdenum.....	0.37	0.40	Chromel-P (90 Ni-10 Cr).....	0.35	
Nickel.....	0.36	0.37	80 Ni-20 Cr.....	0.35	
Palladium.....	0.33	0.37	60 Ni-24 Fe-16 Cr.....	0.36	
Platinum.....	0.30	0.38	Alumel (95 Ni; bal. Al, Mn, Si).....	0.37	
Rhodium.....	0.24	0.30	90 Pt-10 Rh.....	0.27	
Silver.....	0.07	0.07			
Tantalum.....	0.49				

that the corrections necessary to add to the observed temperatures to convert them into true temperatures are small. Thus with iron oxide, the correction is only 10° at 1200°C. The corrections are very large for clear molten metals, but are smaller for the oxides that soon form on the molten surface when exposed to the air. Table 8 shows the true temperatures corresponding to the temperatures observed when sighting on certain materials in the open. For temperature control, it is unnecessary to apply these corrections. The observed temperatures, although known to be low, will be low by the same amount from time to time and hence will serve just as satisfactorily for reproducing temperature conditions in any process as the corrected temperatures. The above statement must be modified if factors other than emissivity of the material require consideration. For example, reproducible results cannot be expected if heavy clouds of smoke are in the line of sight one day and not on the next day. If the pyrometer is sighted on a stream of molten iron during pouring or tapping, the surface of the metal is usually clear and free from oxide. If the stream should

TABLE 10.—SPECTRAL EMISSIVITY OF OXIDES¹

Material	Range of observed values, $\epsilon_{0.65\mu}$	Probable value for the oxide formed on smooth metal, $\epsilon_{0.65\mu}$
Aluminum oxide.....	0.22–0.40	0.30
Beryllium oxide.....	0.07–0.37	0.35
Cerium oxide.....	0.58–0.80	
Chromium oxide.....	0.60–0.80	0.70
Cobalt oxide.....		0.75
Columbium oxide.....	0.55–0.71	0.70
Copper oxide.....	0.60–0.80	0.70
Iron oxide.....	0.63–0.98	0.70
Magnesium oxide.....	0.10–0.43	0.20
Nickel oxide.....	0.85–0.96	0.90
Thorium oxide.....	0.20–0.57	0.50
Tin oxide.....	0.32–0.60	
Titanium oxide.....		0.50
Uranium oxide.....		0.30
Vanadium oxide.....		0.70
Yttrium oxide.....		0.60
Zirconium oxide.....	0.18–0.43	0.40
Alumel (oxidized).....		0.87
Cast iron (oxidized).....		0.70
Chromel-P (90 Ni–10 Cr) (oxidized).....		0.87
80 Ni–20 Cr (oxidized).....		0.90
60 Ni–24 Fe–16 Cr (oxidized).....		0.83
55 Fe–37.5 Cr–7.5 Al (oxidized).....		0.78
70 Fe–23 Cr–5 Al–2 Co (oxidized).....		0.75
Constantan (55 Cu–45 Ni) (oxidized).....		0.84
Carbon steel (oxidized).....		0.80
Stainless steel (18-8) (oxidized).....		0.85
Porcelain.....	0.25–0.50	

¹ The emissivity of oxides and oxidized metals depends to a large extent upon the roughness of the surface. In general, higher values of emissivity are obtained on the rougher surfaces.

at any time contain much slag, the surface will show bright patches on account of the higher emissivity of the slag. To make the readings conform with those taken on the clear stream, one must sight on the darker spaces between the slag or sight upon the slag and correct both sets of data according to Table 8. Tables 9 and 10 give the emissivity of various materials for wave length $\lambda = 0.65$ micron. The change of emissivity with temperature is usually small for metals.

Table 11 shows the corrections that must be added to the readings obtained with an optical pyrometer using light of wave length $\lambda = 0.65$ micron, for various emissivities, in order to obtain the true temperatures. These data are especially useful when plotted with observed temperatures as abscissas and corrections as ordinates. A family of curves is thus obtained corresponding to the different values of the emissivity.

TABLE 11.—CORRECTIONS TO OBSERVED TEMPERATURES FOR PYROMETER USING RED LIGHT
($\lambda = 0.65$ micron, $C_2 = 14,320$ micron-degrees)

Emissivity	Add corrections below for the following observed temperatures, °C.										
	700	800	900	1000	1100	1200	1300	1400	1600	1800	2000
0.30	55	67	80	95	111	129	148	168	214	265	322
0.40	41	50	60	71	83	96	110	125	158	196	237
0.50	31	37	45	53	62	72	82	93	117	145	175
0.60	22	27	33	39	45	52	60	67	85	105	126
0.70	16	19	23	27	31	36	41	47	59	72	87
0.80	10	12	14	17	19	22	25	29	36	44	54
0.90	4	5	7	8	9	10	12	13	17	21	25
1.00	0	0	0	0	0	0	0	0	0	0	0

Use of a Window.—It is frequently necessary, especially in the laboratory, to sight an optical pyrometer into a furnace through a window. What correction must be applied to the observed temperatures to take account of the loss of light at the window? Kanolt has measured the transmission coefficient for a number of ordinary glass windows at $\lambda = 0.65$ micron and obtained a mean value of 0.904. Hence

$$= 0.0000046$$

where ϑ is the true absolute temperature of the source and S is the observed absolute temperature. Table 12 is computed from the above formula.

TABLE 12.—CORRECTION TO OBSERVED TEMPERATURES FOR ABSORPTION OF RED LIGHT ($\lambda = 0.65$ MICRON) BY A SINGLE CLEAN WINDOW

Observed temperatures, °C.	Correction to add, °C.	Observed temperatures, °C.	Correction to add, °C.
600	3.5	1600	16.0
800	5.4	1800	20.0
1000	7.5	2000	24.0
1200	10.0	2500	36.0
1400	13.0	3000	50.0

Flames and Smoke.—The optical pyrometer cannot be used satisfactorily when sighted through flames or smoke. Usually, the presence of luminous flames increases the temperature reading, and the presence of smoke clouds absorbs so much radiation that the pyrometer may read several hundred degrees low. The optical pyrometer can be used to measure the temperature of the slag in an openhearth furnace, but the flames prove a serious hindrance except during reversals when observations may be taken to advantage. In a cement kiln the dust, smoke, and flames all combine to make the observations very untrustworthy. Carbon dioxide, water vapor, and other invisible gases produce no effect.

Radiation Pyrometry.—A radiation pyrometer may be defined as a pyrometer in which the heating effect of the radiant energy falling on a temperature-sensitive element is utilized in the measurement of temperatures. The band of wave lengths used may be of any width and in any part of the spectrum. Commercial instruments of this type ordinarily utilize the entire spectrum of the source at which the instrument is sighted, with the exception of that excluded by lenses and windows when these are used. The Stefan-Boltzmann law expressing the relation between the intensity of energy J (of all wave lengths) radiated by a black body and its absolute temperature ϑ is as follows:

$$J = \sigma(\vartheta^4 - \vartheta_0^4) \quad (1)$$

where ϑ_0 denotes the absolute temperature of the surroundings or of the measuring instrument receiving the radiation, and σ an empirical constant (in industrial instruments). Usually ϑ_0^4 is negligible in comparison with ϑ^4 , and Eq. (1) reduces to

$$J = \sigma\vartheta^4 \text{ for a black body} \quad (2)$$

For a nonblack body,

$$J' = \sigma E\vartheta^4 = \sigma S^4 \quad (3)$$

where E is the total emissivity and S the apparent absolute temperature of the object sighted upon as measured by the radiation pyrometer. From Eq. (3)

$$E = \frac{S^4}{\vartheta^4} \quad \text{or} \quad \log E = 4(\log S - \log \vartheta) \quad (4)$$

Thus, knowing the total emissivity E of any material, it is possible to obtain the true temperature ϑ from the apparent temperature S measured by a radiation pyrometer.

As will be explained later, none of the above equations applies exactly for any actual radiation pyrometer, and calibration at more than one temperature is necessary.

An optical pyrometer measures the intensity of a narrow spectral band of radiation emitted by a glowing object. The radiation pyrometer measures the intensity of practically all wave lengths, the light rays and the heat rays combined. Usually, the energy of all wave lengths radiated by the source is focused in some manner upon the hot junctions of several thermocouples in series. The temperature to which these junctions rise is approximately proportional to the rate at which energy falls upon them, which, in turn, by the Stefan-Boltzmann law, is proportional to the fourth power of the absolute temperature of the source. The rise in temperature of the hot junctions of the thermopile generates a thermoelectric e.m.f. Hence, the calibration of a radiation pyrometer consists in determining the relation between the e.m.f. developed and the temperature of the source sighted upon. Most commercial radiation pyrometers employ a lens to focus the energy upon the thermopile. In some instances where other methods are employed to focus the energy, the entrance to the pyrometer is closed with a quartz, pyrex, or mica window to exclude dust and fumes. In either case, the lens or the window excludes certain of the long wave lengths.

Consequently, the energy received by the thermopile is not exactly proportional to the fourth power of the absolute temperature. The energy received at the lower temperatures may vary as the fifth or sixth power of the absolute temperature, but it approaches the fourth power at the higher temperatures. This is of no serious consequence because radiation pyrometers are usually calibrated by comparison with an optical pyrometer or a thermocouple. The e.m.f. may be measured by a potentiometer or galvanometer, or by any of the methods discussed under thermoelectric pyrometry. The same type of instrument is used with the radiation pyrometer as with the ordinary thermocouple. In fact, the entire discussion on methods of measuring the e.m.f. of thermocouples is equally applicable to radiation pyrometry. The temperature of the cold junction of the couple in the radiation pyrometer is not controlled. The hot and cold junctions are in fairly close proximity and are, hence, equally affected by changes in room temperature. The cold junction is always shaded from the heat radiated by the source sighted upon.

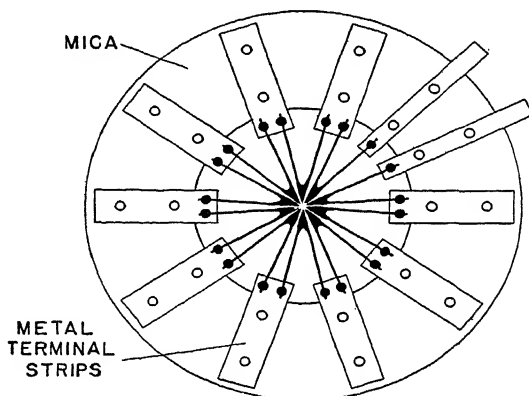


FIG. 13.—Thermopile of a radiation pyrometer.

The essential elements of a radiation pyrometer receiver are a thermopile and a means of focusing the energy on the thermopile, enclosed in a telescope. The structure of one form of thermopile is shown in Fig. 13. The number of thermocouples in thermopiles ranges from about four to ten. The hot junctions of the thermocouples are flattened and blackened on the exposed side to absorb the incident energy. The cold junctions are attached to metal strips which are insulated with annular mica disks and clamped in the thermopile housing.

Commercial radiation receivers are ordinarily of the fixed-focus type and are designed to be used at distances from the source greater than a certain minimum value. The source sighted upon should be large enough to satisfy the optical requirements of the receiver. This can be determined from data supplied with the receiver and verified by sighting through the sight window usually provided for this purpose. The hot junctions of the thermopile that are seen through the sight window should be completely covered by the image of the source.

The maximum permissible ambient temperature ranges from 150 to 250°F. If the radiation receiver is exposed to temperatures higher than the permissible value, means should be provided for cooling. The instrument companies have cooling jackets available.

Commercial radiation receivers are usually provided with such features as diaphragms to shield the cold junction from radiation, compensating coils to minimize the effects of changes in ambient temperature, and calibration-adjustment devices.

Radiation pyrometer receivers may be divided into three general classes, depending on the method employed to focus the radiant energy on the sensitive element. These are the lens type, the mirror type, and the cone type.

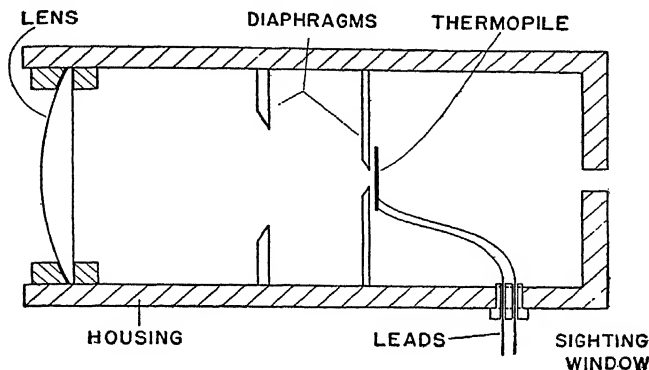


FIG. 14.—Lens type of radiation receiver.

Figure 14 shows the essential elements of a lens-type receiver, which is the one most generally used for high temperatures. Instruments of this type are not suitable for temperatures below about 800°F. because practically all the energy at temperatures lower than this is excluded by the lens. The ratio of the diameter of the source to the distance from the lens to the source ranges from about 1:20 in some instruments to about 1:7 in others. The minimum diameter of source ranges from about 1 to 2 in.

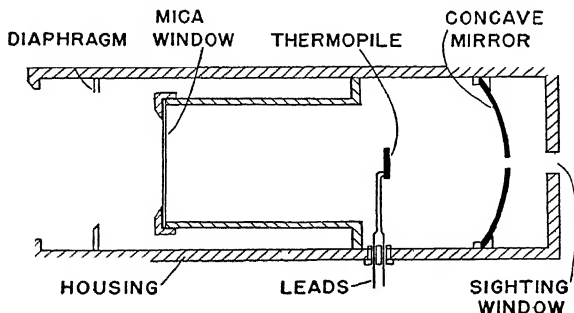


FIG. 15.—Mirror type of radiation receiver.

Figure 15 shows the arrangement of the essential elements of the mirror-type radiation receiver. The opening in the telescope is usually closed with a thin mica window to exclude dust and fumes. Commercial instruments of this type are suitable for temperatures down to about 225°F. In order to get sufficient energy to give satisfactory results at the low temperatures, the field of view is made larger than that of the lens type used at the higher temperatures.

The cone-type radiation receiver is shown diagrammatically in Fig. 16. Radiant energy entering the diaphragm *A* falls upon the conical mirror *K* and by multiple reflection is concentrated on the thermopile *C*.

Sources of Error in Radiation Pyrometers.—The most common source of error is caused by dirt and dust collecting on the surfaces of lenses, windows, and reflecting surfaces. All industrial types of radiation pyrometer tend to read low, the greater the sighting distance and the smaller the source. This is not due primarily to the absorption of heat by carbon dioxide and water vapor (a small effect at short distances), but to the imperfect limitation of the cone or cross-sectional area of the rays falling upon the receiver, and the imperfect limitation of the area of the receiver, which may be more exactly stated as an imperfect limitation of the quantity of heat absorbed by the receiver. There may be reflection from the walls of the receiver tube upon the receiver, absorption of heat by the walls and limiting diaphragms, and reradiation of the absorbed heat to the receiver or thermocouple. This latter effect is practically eliminated in the more recent designs by the use of cooling jackets, compensators, or both.

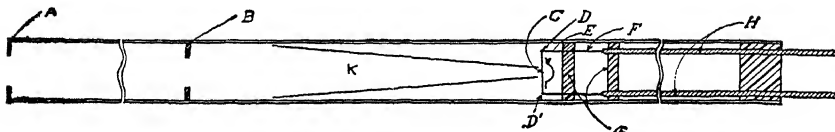


Fig. 16.—Thwing radiation pyrometer.

Advantages and Disadvantages of Radiation Pyrometers.—For temperatures above 1400 or 1500°C., either a radiation pyrometer or an optical pyrometer is usually employed. The optical pyrometer is capable of higher accuracy and is less susceptible to errors than the radiation pyrometer. Smoke and dust affect the readings of both instruments, but the radiation pyrometer is affected by the presence of cooler strata of carbon dioxide and other gaseous combustion products in the furnace. Carbon dioxide and water vapor absorb the heat rays, and hence the radiation pyrometer will read too low when sighted through such gases. The main advantage of the radiation pyrometer is the fact that it is easily made automatically recording. The recording mechanism is the same as that employed for ordinary thermocouples, discussed in the section on Recording Pyrometry. The radiation pyrometer is desirable for many processes of lower temperature where thermocouples cannot be conveniently installed. It is also useful in measuring the surface temperature of large ingots, sheet, strip, rails, etc.

Black-body and Nonblack-body Conditions.—Radiation pyrometers are calibrated to read correctly when sighted on a black body. Most furnaces approximate black-body conditions sufficiently well. When sighted on materials in the open, certain corrections must be applied to the observed temperature to convert them into true temperatures. In case the temperature of one section of an unequally heated furnace is required or if the furnace contains much smoke or dust, it is frequently desirable to sight the radiation pyrometer into the bottom of a fire-clay or porcelain tube as illustrated by Fig. 17. The tube should be uniformly heated at the end for a length at least three times its diameter. The tube must have a diameter such that the cone of rays entering the telescope is not intercepted by the front of the tube. Figure 18 illustrates this condition. A usual practice is to mount the radiation receiver 24 in. from the bottom of a tube 2 in. in diameter.

Table 13 shows the true temperatures corresponding to the apparent or measured temperatures observed with a radiation pyrometer when sighted on various materials

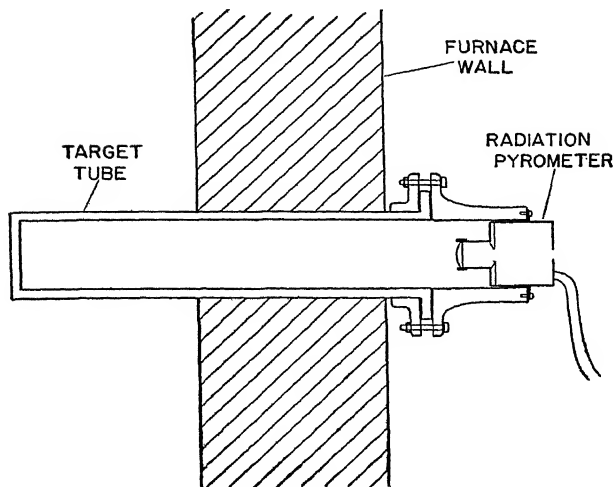


FIG. 17.—Use of closed end tube with radiation receiver.

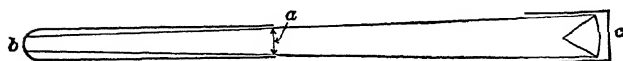


FIG. 18.—Porcelain tube for observing temperatures.

TABLE 13.—TRUE VS. APPARENT TEMPERATURES MEASURED BY RADIATION PYROMETERS WHEN SIGHTED ON MATERIALS IN THE OPEN

Observed temperature, °C.	True temperature, °C.				
	Molten iron	Molten copper	Copper oxide	Iron oxide	Nickel oxide
600	1130	720	630	710
650	1210	775	755
700	1290	830	735	800
750	890	845
800	1200	945	840	895
850	1270	1000	940
900	1340	1060	945	985
950	1410	1115	1030
1000	1475	1170	1050	1075
1050	1550	1120
1100	1610	1155	1165
1150	1680	1210
1200	1750	1260	1255

in the open. This table must not be confused with Table 8 for the optical pyrometer. It will be noticed that the corrections are entirely different for the two types of pyrometer. Radiation pyrometers have not been found to be very useful for measuring temperatures of nonblack bodies.

Resistance Thermometry.[14]—The operation of a resistance thermometer depends on the variation with temperature of the resistance of an electrical conductor. A high-temperature coefficient of resistivity might, accordingly, appear of primary importance. Certain oxides possess this property, having a resistance at room temperature from a hundred thousand to a million times that at 1000°C. However, the resistance of a suitably mounted material may be easily and accurately determined, so that the consideration of high-temperature coefficient is rarely, if ever, the deciding factor in the choice of the material. The fundamentally important property is reproducibility and constancy of resistance. A further desirable feature is a simple and convenient relation between temperature and resistance. Platinum of the highest obtainable purity meets these requirements up to about 1100°C. The temperature coefficient of resistance of platinum is so well established that a working temperature scale up to 1083°C. may be based on this form of pyrometer calibrated according to certain specifications. On the basis of this reproducible working temperature scale, temperatures may be measured to about 0.1°C. at 1000°C. To obtain such precision, however, requires accurately constructed apparatus and very careful experimental manipulation.

For industrial purposes, resistance thermometers are made usually either of copper, platinum, or nickel, the latter material being satisfactory for temperatures below 300°C. Although the resistance thermometer is of the greatest importance in the laboratory for work of extremely high precision, its importance for industrial processes at high temperatures is limited. The precision of which the instrument is capable requires many precautions easily overlooked. Bad contacts in the line, a broken strand in a flexible cable, etc., may cause large errors. The thermometer is fragile, and its calibration may be altered by severe handling, by the slightest contamination, or by volatilization of the platinum wire forming the coil. The range is confined usually below about 1000°C., whereas rare-metal thermocouples may be used to 1400°C.

In the early development of pyrometry before satisfactory galvanometers were devised for thermocouples, the resistance thermometer afforded about the only accurate means of temperature measurement, but with the highly sensitive and reliable thermocouple indicators and portable potentiometers and with the reproducible platinum to platinum-rhodium couple now obtainable, few industrial processes are carried out above 300°C. where a thermoelectric installation is not to be preferred. When very small temperature changes or intervals must be measured with high precision, the resistance thermometer may be employed effectively. Also in certain processes where an integrated or average temperature over the entire bulb is required, the resistance thermometer is highly desirable. A homogeneous thermocouple indicates the temperature of its hot junction, while a resistance thermometer indicates the average temperature of the entire coil or bulb, which may be constructed as long as desired. There are no cold-junction corrections for the resistance thermometer—a decided advantage over the thermocouple, but one that is offset by the disadvantages mentioned.

Recording Pyrometry.—The pyrometers that can be easily made to record automatically fall under the following classifications; (1) gas, saturated vapor, and liquid thermometers: (2) resistance thermometers: (3) thermoelectric pyrometers: (4) radiation pyrometers.

Of these four types, the thermoelectric pyrometer recorder has the greatest applicability, especially for the higher temperatures at which the first two named are not suitable. The constant-volume industrial gas thermometer is successful up to about 400°C. The resistance thermometer is capable of high accuracy up to 1000°C. At such high temperatures, however, thermocouples are more serviceable, since deterioration of the wire from continual heating does not so seriously alter the e.m.f. developed by a couple as it does the resistance of a resistance thermometer. Radiation pyrometers are useful at the highest attainable temperatures, but processes in which extremely high temperatures are employed are not, in general, subjected to precise temperature control.

The most usual form of temperature record is that in which temperature appears as one coordinate and time as the other coordinate. Such a temperature-time curve has been called an "autographic record." This type of curve is valuable as a continuous record of the temperature of a furnace over an extended interval of time. It may be employed to detect transformation points in steel, which appear as flexures or indentations on the plot when the furnace containing the sample of steel is uniformly heated or cooled. For this latter work, however, the "differential-temperature" curve recorder is especially adapted and will be described later. The various special methods have been discussed by Burgess[15].

Potentiometric Recorders.—Recorders of this type are made by a number of companies and are available with practically any feature desired, such as circular or strip chart, single-point or multiple-point, calibrated in e.m.f. or directly in degrees centigrade or Fahrenheit for practically any useful temperature range of any type of thermocouple or radiation receiver, with or without reference-junction compensation, various chart speeds, with or without automatic control features, automatic or manual adjustment of battery current, automatic recording of battery current, etc. The widths of strip charts of recorders of this type ranges from 10 to 12 in. This type of instrument may be divided into the following three general classes, depending upon the method used to detect the amount of unbalance of the potentiometer circuit and to return the circuit to balance: the mechanical type, the photoelectric type, and the electronic type.

The mechanical type of potentiometer recorder is, by far, the most commonly used at the present time. It consists essentially of a potentiometer with a mechanical movement for detecting the position of a galvanometer pointer and automatically moving a carriage carrying the pen or print wheel across the paper and simultaneously changing the slide-wire contact to a balanced position. These moving parts as well as the paper chart are driven by an electric motor. Synchronous motors are ordinarily used to obtain uniform chart speed.

In the phototube type of recorder, a mirror-type galvanometer is used to reflect a light beam to a phototube. This beam of light in moving on and off the phototube passes the controlling edge of a screen, causing relays to operate which, in turn, control a reversing motor that drives the slide-wire contact of the potentiometer. The phototube is not a calibrated element but serves only to detect the direction of the light-beam and to bring the potentiometer to a balance.

In the electronic type of recorder, the conventional galvanometer and its associated mechanisms are eliminated. In one form of instrument of this type, the unbalanced d-c voltage between the e.m.f. of the thermocouple and the opposing potential of the slide wire is converted into an alternating voltage by a conversion stage, then amplified and fed into one winding of a two-phase induction motor. The other winding of this motor is continuously energized from the power mains, and accordingly the direction of rotation is governed by the phase of the voltage delivered by the conversion stage.

Since the phase of this voltage reverses with reversal of the direction in which the potentiometer is out of balance, the motor will rotate, when required, always in the proper direction to restore the condition of balance.

Instruments of this type can be classified as extremely high-speed recorders. Inasmuch as they have no galvanometer, or other delicate parts, they are rugged and unaffected by vibration.

Galvanometric Recorders.—Circular-chart and roll-chart recorders of this type have been available, but they are rapidly being replaced by recorders of the potentiometric type. The principal reason for this is that the recorders of the galvanometric type do not give the accuracy required in most cases.

Transformation-point Indicators and Recorders.—Instruments for obtaining transformation or critical points in steel have been in use for a number of years and are fairly well known in the steel industries. The simplest and also the least accurate and sensitive method of measuring the temperature at critical points is to record or plot from indicator readings the temperature-time curve of the sample of steel when placed in a furnace and heated. This method requires a steady rate of heating, and the sensitivity is ordinarily only sufficient for high-carbon steels. It is possible, by using a potentiometer and a sensitive galvanometer and forcing the furnace to rise in temperature at a steady rate, to obtain a temperature-time curve that will plainly indicate the transformations. In fact, this method is used for treating small parts, such as gears and tools. A more accurate method for studying or testing steels is the inverse-rate method developed at the National Bureau of Standards[16].

Usually, ample sensitivity can be obtained by the inverse-rate method, but if a special study is being made, and all possible sensitivity is desired, the method of a neutral body and differential couple, devised by Roberts-Austen, can be resorted to, in which the data appear as temperature versus difference in temperature of test piece and neutral body. These data can be replotted as

$$T \text{ vs. } \frac{\Delta(T - T')}{T}$$

the so-called derived-differential curve due to Rosenhain[17], especially if errors are suspected as a result of considerable differences in the cooling curves of the test piece and neutral body. This curve will always aid in the interpretation of results, and corresponds to the inverse-rate curve obtained from measurements of time and temperature only, on a test piece without the neutral body and differential thermocouple.

Recorders for transformation points require no special description in so brief a treatment. Ordinary recorders suffice for temperature-time curves and can be easily applied to the differential method mentioned in the previous paragraph. The inverse-rate curve has been obtained at the National Bureau of Standards by an automatic device developed by French[18]. This device eliminates the tedious work of plotting data point by point, but is too complex for general use.

High-temperature Control.—The meaning of temperature control can be extended to cover not only the control of a furnace to a certain temperature, but also the control of a process by means of temperature measurements and the knowledge they furnish.

Some of the factors that increase the difficulty of the regulation of furnaces, ovens, etc., are inconstancy of heat supply, variation in internal absorption or generation of heat, variation of loss of heat by radiation, etc., and unsteady supply or composition of material to be heat-treated. Control of a furnace is best accomplished by use of a pyrometer. There is always some point, or perhaps several, in a furnace, the temperature of which is more intimately connected with the desired conditions than any other. In a hardening furnace, it is a simple matter to put a thermocouple at some

point and find the temperature in the immediate vicinity. But it is another matter to determine the actual temperature of a piece of steel in the furnace. The control of a furnace is more difficult in the absence of some physical change taking place, which may serve even as an accurate pyrometer in some cases. There are, in general, two points or classes of points, the temperature of which are especially important. One of these is at the object or material being heated, and the other is at the point or part of the furnace where the widest variations in temperature occur. It is often advantageous to control the furnace from measurements at the latter point, while adjusting the control at intervals according to measurements at the former point. It is frequently impossible or impracticable, however, to obtain pyrometric measurements at the point where the widest variations in temperature occur. It is, nevertheless, important to consider these facts. If thermocouples are used, they must be placed where the indicated temperature will have direct relation to the desired conditions.

Automatic Temperature Control.—The general principle of automatic control is to allow the furnace to heat and cool over a limited range and, by means of the resulting variations in e.m.f. of a thermocouple, radiation pyrometer, etc., to apply corrective changes in the heat supply. Such control is complicated by the interrelation of such factors as thermal lag and the magnitude of the temperature variations during each period of heating and cooling. The actual variations in the temperature of different parts of the furnace will, in general, be greater than the variations indicated by the pyrometer. This is unavoidable. The best results are obtained by such adjustments at intervals as will result in the smallest possible variation in the indicated temperatures. The furnace operator, in such case, must correct any wide, erratic, or sudden changes in fuel or heat supply or supply of raw material.

Automatic temperature controllers are available in as many or more forms than recorders discussed earlier. These have so developed that it is now possible to control automatically the heating and cooling rates, as well as the regulation of the "soaking" temperature. No attempt will be made to discuss the various possibilities here because they appear to be unlimited.

Calibration of Pyrometers.—There are innumerable methods of standardizing, calibrating, or checking pyrometers. The choice in any instance depends upon the type of pyrometer, temperature range, accuracy required, apparatus available, personal preference, etc. Only a few of these methods will be discussed here, because of limited space.

The accuracy required in pyrometer calibrations ranges from 0.01° in certain laboratory work to 10 or 20°C. in certain industrial applications. For an accuracy of 0.1°C. or better, agreement with the International Temperature Scale and methods of interpolating between calibration points become problems of prime importance, but for an accuracy of about 10°C. , comparatively simple methods of calibration will usually suffice.

The methods of calibrating pyrometers may be divided into two general classes, depending on the method of determining the temperature: (1) primary calibrations in which fixed temperatures, defined by the melting, freezing, or boiling points of substances are used; and (2) secondary calibrations in which another pyrometer is used as a standard of reference or comparison. In general, the secondary, or comparison, method is preferred for industrial laboratories. In such cases, the standard pyrometers should be checked regularly by the National Bureau of Standards or some other laboratory equipped to carry out such standardization. The National Bureau of Standards undertakes the calibration of comparison standards upon application. Many industrial laboratories, however, carry out their own primary calibrations. For this purpose, the National Bureau of Standards supplies standard freezing-point samples of copper, aluminum, zinc, lead, and tin.

Calibration of High-temperature Thermometers.—Thermometers can be calibrated by comparison with standards in a water bath below 100°C., in an oil bath from 100 to 300° or even 400°C., and in a tin, lead, or fused-salt bath to 550°C. The baths should be well stirred.

Fixed points to be used are 0°, 100°, and the boiling points of naphthalene, 218°C.; benzophenone, 306°C.; anthracene, 340°C.; and sulphur, 444.6°C. The boiling-point method is very inconvenient and, in general, more subject to error than the stirred-bath comparison method. In checking thermometers, special attention must be given to (1) uniformity of temperature of bath, (2) stem corrections of thermometers, (3) change in the ice-point reading, and (4) use of the thermometers installed with short immersion.

Calibration of Thermocouples.—Detailed descriptions of methods of calibrating thermocouples are given in a reference [19] at the end of the chapter.

It is the practice at the National Bureau of Standards, at the present time, to remove platinum to platinum-rhodium thermocouples from their insulating and protecting tubes and heat them in air for 1 hr. at about 1450°C. before calibrating. There is some question as to whether this is necessary or beneficial for new thermocouple wire because some of the platinum-thermocouple manufacturers give their wire a stabilizing treatment. Practically all base-metal thermocouple wire produced in this country is annealed or given a "stabilizing heat-treatment" by the manufacturer.

In general, considerable time is required to calibrate a thermocouple at the freezing points of metals. Except in special cases, this method is limited to the calibration of primary platinum to 90 platinum-10 rhodium thermocouple standards. Occasionally it may be used advantageously to check a thermocouple of any type at a few selected points.

In calibrating a thermocouple at the freezing point of a metal, the general procedure is to immerse the thermocouple, properly protected, in the molten metal and take simultaneous readings of e.m.f. and time as the metal cools slowly through its freezing point. Precautions must be taken to assure that the depth of immersion is sufficient to prevent heating or cooling of the junction by heat flow along the thermocouple wires and protection tube. The depth required will depend on the material and size of the wires, the dimensions of the insulating and protecting tubes, and the difference between the temperature of the freezing-point sample and that of the furnace and atmosphere immediately above it. The safest method of determining whether the depth of immersion is sufficient is by trial. It should be such that during the period of freezing the thermocouple can be lowered or raised at least 1 cm. from its normal position without altering the indicated e.m.f. by as much as the allowable uncertainty in the calibration.

Calibration by Comparison Methods.—The calibration of a thermocouple by comparison with a working standard is sufficiently accurate for most purposes and can be done conveniently in most industrial and technical laboratories. The success of this method usually depends on the ability of the observer to bring the junction of the couple to the same temperature as the actuating element of the standard, such as the hot junction of a standard thermocouple or the bulb of a resistance or liquid-in-glass thermometer. The accuracy obtained is further limited by the accuracy of the standard. Of course, the reference-junction temperature must be known, but this can usually be controlled by using an ice bath of water and cracked ice or measured by a liquid-in-glass thermometer. The method of bringing the junction of the couple to the same temperature as that of the actuating element of the standard depends upon the type of couple, type of standard, and the method of heating.

In the comparison of platinum to platinum-rhodium thermocouples, the junctions may be welded together or wrapped together with platinum wire or foil and heated

by any convenient means. The thermocouple wires should be protected from metallic vapors and products of combustion if a fuel-fired furnace is used.

In the testing of one or more small base-metal thermocouples, they may be welded to the junction of the standard. If a large number of base-metal thermocouples are to be tested at the same temperature, the method of immersing the thermocouples in a molten-metal bath or into holes drilled in a large copper or aluminum block is very advantageous. If a tin bath is used, iron or nickel-chromium tubes are sufficient protection for base-metal thermocouples. When wires, insulators, and protection tubes of base-metal thermocouples are large, tests should be made to ensure that the depth of immersion is sufficient to eliminate heating or cooling of the junction by heat flow along these materials.

Satisfactory results are obtained by purchasing base-metal thermocouple wire in sufficient quantities to make a large number of thermocouples, calibrating a few sample thermocouples from the coil, and applying this calibration to the other thermocouples made from the same coil.

Thermocouples in Fixed Installations.—After thermocouples have been used for some time at high temperatures, it is difficult if not impossible to determine how much the calibrations are in error by removing them from an installation and testing in a laboratory furnace. The thermocouples are usually inhomogeneous after such use and in such a condition that the e.m.f. developed by the couples depends on the temperature distribution along the wires. If possible, such couples should be tested under the same conditions and in the same installation in which they are used. Although it is not usually possible to obtain as high a precision by testing the couples in place as is obtained in laboratory tests, the results are far more accurate in the sense of being representative of the behavior of the couples.

The exact method of procedure depends on the type of installation. Usually it is possible to place the standard beside the thermocouple being tested. Frequently, there is room to insert a small standard thermocouple in the protection tube alongside of the thermocouple being tested. In any case, the hot junction ends of the thermocouples should be brought as close together as possible.

Calibration of Indicators and Recorders.—The calibration of indicators and recorders in industrial laboratories or plants will generally consist of a secondary calibration, *i.e.*, comparing one instrument with another, which is referred to as a standard.

In calibrating an indicator of the galvanometer type, a millivoltmeter, it should be borne in mind that the deflection of the pointer of the instrument is proportional to the current through the moving coil (considering torque of springs proportional to such deflection and magnetic field uniform). Consequently, the resistance of the thermocouple and leads must be taken into consideration. This may be done by (1) connecting the thermocouple and leads in series with the instrument during calibration; (2) connecting a resistance equal to that of the thermocouple and leads in series with the indicator; or (3) measuring the resistance of the thermocouple, leads, and instrument and making the necessary corrections.

Indicators having scales reading only in temperature must be calibrated while connected to a thermocouple for which the instrument was designed, or else the temperature-e.m.f. relation must be known. Careful attention to reference junctions, their disposition and compensation, should be given.

Potentiometers are compared by attaching their terminals together + to + and - to - and balancing each.

Calibration of Optical Pyrometers.—The primary standardization of an optical pyrometer by sighting into a black body immersed in freezing or melting metal is difficult and tedious. A secondary calibration by comparing two instruments is fairly

simple. The two opticals may be sighted alternately or simultaneously on the same spot of any convenient source. Tungsten-filament strip lamps are available for carrying out such comparisons in the laboratory. These lamps usually are provided with a notch in the filament to permit sighting the two opticals at the same spot.

Strip lamps may also provide a convenient means of checking an optical pyrometer when a working standard is not available. This is accomplished by first determining (or having determined) corresponding values of temperature and current through the strip lamp and using it as a standard.

Facilities for controlling the current through the lamp are required in either case.

The calibration of optical pyrometers and strip lamps is carried out at the National Bureau of Standards on application.

Calibration of Radiation Pyrometers.—The best results on the calibration of a radiation pyrometer are usually obtained by calibrating it (1) after it has been installed or (2) at a fixed distance from a definite target. A thermocouple or, preferably, an optical pyrometer may be used as a standard. The optical pyrometer should be sighted at the same point as the radiation pyrometer. If the object that the radiation pyrometer is sighted on does not very closely approach a black body, correction must be applied for the emissivity.

The practice of calibrating a radiation pyrometer under one set of conditions in regard to distance from target, size of target, type of black body, etc., and then using it under another set of conditions is subject to some uncertainty. The primary difficulty is the construction of a uniformly heated black body with an opening sufficiently large to meet the requirements of the radiation pyrometer. If such a calibration is attempted in a laboratory, care should be taken to see that the distance and size of opening are within the limits set by the manufacturer.

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CHAPTER X

ROASTING AND SINTERING

BY ARTHUR S. DWIGHT¹

Introduction.—When a metallic sulphide ore in the natural state is gradually heated in the presence of atmospheric oxygen, the following succession of phenomena will be observed as the temperature rises: (1) The mechanically held moisture will be expelled, and up to 100°C. at normal pressure no other important change will take place. This stage is called “drying.” (2) As the temperature rises above 100°C., other compounds will be expelled, such as chemically combined water, volatile carbonaceous matter, and carbonic acid from carbonates, but, through a considerable range of temperature, without the absorption of oxygen by the metallic elements. This stage is called “calcination,” and the most common example is the making of quicklime from limestone by burning. (3) As the temperature approaches a low red heat, the metallic sulphides begin to be attacked by the oxygen of the air with the formation of oxides and sulphates of the metals, and sulphurous acid gas (SO₂). This stage is called “roasting.” (4) The sulphide minerals are mostly relatively fusible: hence, if the temperature is raised too rapidly (after roasting begins), some of the unroasted sulphide will fuse together or “matte,” and the roasting operation will be interfered with—perhaps stopped altogether. This premature melting is sometimes erroneously called “sintering.” If, on the other hand, a favorable roasting temperature is maintained, with ample contact of the material with atmospheric oxygen, the mineral will be rapidly oxidized, the reaction being propagated from the exterior to the interior of the mineral mass, but without fusion.

Therefore, roasting may be defined as a metallurgical operation wherein an ore is heated under such conditions and to such a temperature (below that of fusion of its mineral constituents) that the metal component of the ore which it is sought to recover will be chemically changed from that form in which it occurs in the ore to some other form which will be amenable to some definite subsequent treatment.

The roasting process may be any one of the following kinds: (1) oxidizing roast, (2) reducing roast, (3) sulphating roast, (4) chloridizing roast.

Certain effects or preliminary reactions are common to all types of roast, the first being dehydration, both of moisture and of water of crystallization, and also, with carbonates, the driving off of the CO₂, which may be expressed by such a general formula as



Size of material for roasting should be governed by the manner of roasting to be employed (or vice versa) or by the subsequent treatment for which the roasted ore is destined: it may vary from a diameter of several inches in the weathering process, as at Rio Tinto—or even the roasting of pyrite ore “in place” suggested

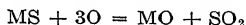
¹ President, Dwight & Lloyd Sintering Co., New York. The author acknowledges his indebtedness to Robert C. Canby, Reed W. Hyde, and Robert W. Rowen for their cooperation and assistance in the preparation of this chapter.

by and experiments made with idea of following by leaching in place¹—down to the pulverization of zinc ores for the dead roasting required for subsequent retorting (0.04 to 0.08 in.).²

The best roasting experience has shown that the "sizing" of the crushed ore into several different sized products would result in a much better roaster practice, but there are, apparently, sufficient reasons, such as limited tonnage, for not adopting the practice. The more recent advent of "blast roasting," however, would now seem to render unnecessary such sizing in smelter practice, but for hydrometallurgical roasting, where tonnages permit, it is well worthy of consideration, as each size could then receive the time and the character of rabbling suitable to that size.

Oxidizing roasting is the kind generally referred to in smelting operations, and it is the roast now most generally practiced in preparing sulphide ores for acid lixiviation, particularly zinc ores for subsequent electrolytic disposition—in which practice the amount of allowable sulphate of zinc is usually limited³—and for zinc retorts, where absolute dead roast is desired.

A general formula for oxidizing roast may be expressed as follows:



Such a general formula, however, cannot be considered as adequately expressing the reactions of the oxidizing roast, but, rather, the ultimate theoretical result, which is arrived at through a complicated series of reversible reactions, which in their progress include the various stages of (2) the reducing roast, and (3) the sulphating roast. The ultimate breaking up of the sulphates, especially of zinc, demand a high degree of temperature in order that the result may be a complete or "dead roast." Neither does such a formula express the influence of catalytic reactions, such as the beneficial effect of a certain proportion of quartz or other minerals having catalyzing properties in the gangue. Time, temperature, oxygen (air), and suitable physical and mechanical conditions are essential elements. Time may vary from the mere flash of the "blast roasting," the hours of the hearth roast, the months of the heap roast, to the years of the "weather roasting" (Rio Tinto practice). This latter practice is not properly considered as under the head of the oxidizing roast.

The temperature for an oxidizing roast must, toward the end of the operation, be sufficient to break up the sulphates that are inevitably formed in the earlier stages of the roast. The temperature depends very largely upon the character of the minerals in the ore, and, while in some cases the temperature may be described as a bright red heat, in zinc ores zinc sulphate gives up only part of its sulphuric acid, forming basic sulphate, which is completely decomposed into zinc oxide, sulphur dioxide, and oxygen only at white heat.⁴ At Anaconda, in oxidizing roasting of 0.573 per cent Cu tailings in six-hearth MacDougall-type roasters, the best results were obtained with the fourth hearth at about 500°C. In the hottest (the third) fired hearth, 535°C., S, 2.2 per cent was reduced to 0.6 per cent.⁵ In roasting for hydrometallurgical purposes, however, other reactions than oxidization, such as formation of ferrites, have to be considered. Some information on this point is given on pages 293 and 296. Broadly, it may be stated that the high temperatures necessary to break up the sulphate tend to cause the formation of ferrites, which are objectionable because of their insolubility in dilute acid.

¹ *Eng. Mining Jour.-Press*, June 21, 1924, p. 1018.

² SCHNABEL, Vol. 2, p. 113.

³ MASON, *Mining Sci. Press*, June 12, 1920, p. 866.

⁴ SCHNABEL, "Handbook of Metallurgy," p. 422; 900 to 970°C., GREENAWALT, "Hydrometallurgy," p. 49.

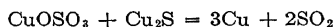
⁵ *Trans. A.I.M.E.*, Vol. 55, p. 186, 1916.

Experiments made at Trail, B.C., indicate that actual physical contact of the iron mineral (pyrite) and zinc mineral is necessary to the formation of zinc ferrite: that, therefore, if the particles were falling freely during the roasting—instead of being in contact as in an ore bed upon a hearth—ferrite would not be formed from such free-falling particles of nonferrous mineral.¹

As oxidizing roasting possesses the major interest to the metallurgist, a more detailed discussion will be found on page 280.

Reducing roasting usually implies a heat-treatment of an oxidized metallic compound under the influence of a so-called reducing agent, usually some active form of hydrogen, carbon, or a metallic sulphide at a temperature below that of fusion, for the purpose of lowering the oxide state or for the complete reduction to metal.

The production of "moss copper" from roasting high-grade matte, as in the Crooke's process, is an instance of reduction to metallic state, the usually assumed reaction being that between sulphate and cuprous sulphide



Another example is furnished by the so-called roast-reaction method of smelting lead ores by the reaction, in a roasting furnace, of PbSO_4 and PbO on unchanged PbS .

Still another example is the production of iron sponge from iron ore mixed with carbon. The Republic Steel Corp. at Warren, Ohio, is installing a 22 ft. 3 in. diameter 12-hearth Nichols-Herreshoff furnace for producing a form of sponge-iron from high-grade eastern magnetite ores. Hot coke-oven gas will be used as the reducing agent and as the means for supplying the heat necessary for this reaction. It is expected that the Fe_3O_4 will be reduced to metallic iron at a temperature of about 1300°F.

The Nicaro Nickel Co. is using 12 Nichols-Herreshoff furnaces 22 ft. 3 in. diameter with 16 hearths. The upper 5 hearths are used for drying and preheating the ore, which is very high in moisture, and the lower 11 hearths serve to reduce partially the iron and nickel present in the mixture of limonite and serpentine ores. The reducing agent is producer gas, and heat is supplied by oil burners that are operated with about 1 per cent CO and with no excess air.

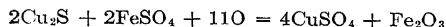
Sulphating roasting occurs incidentally during oxidizing roasting, but is essentially the reverse operation, since the effort is made to produce the maximum amount of sulphate of the metals by coordinating the temperature and the relative proportion of oxygen and SO_2 in the furnace gases. This process will be discussed in its reverse phase under Oxidizing Roasting. The muffle furnace, from its greater possibility of controlling the relative proportions of O, SO_2 , and SO_3 , because the gases of combustion do not come in contact with the roasting ore, has long been believed to be best suited to the sulphating roast.

In more recent reverberatory-furnace practice, however, ingenious schemes of by-passing the material and the gases, and the improved devices for assuring temperature control, are making more promising the economic production of sulphates in this type of furnace, especially in the multiple-hearth furnace. The Coolbaugh process is an instance of withdrawing the gases produced in a preliminary step of the roast and then passing them, with their content of SO_2 , into the atmosphere of another furnace, or that region of the same furnace in which the final reaction of the roast takes place (at a temperature of 450 to 850°C.), an operation that may be arrived at by simply drawing the gases down through the furnace so that the maximum percentage of SO_2 in the furnace gases is present at the final stage of the roast.² Middleton claims to

¹ *Bull. Can. Inst. Min. Met.*, June, 1924, p. 435.

² U.S. patent 1315761.

convert sulphides to sulphates "in any standard type of roasting furnace" by close temperature control. He states that from the reactions of his process "it will be seen that the greater part of the sulphating action is due to the iron sulphate formed during the first stage of the roast, and it has been found that this compound also reacts with oxide of copper, forming copper sulphate."¹ Catalytic action may play an important role here. Some corroboration of this idea may be found in the fact that experience in roasting various zinc ores has shown that those higher in iron produce the greater proportion of zinc sulphate, thus suggesting catalytic action of iron oxide.² The percentage sulphated in Middleton's roast, as shown by sizing tests, rises very sharply at 100 mesh and finer. He suggests the following reaction:



Complete sulphating undoubtedly will be found not wholly to "depend upon temperature control," nor will a theoretically ideal atmosphere alone effect complete sulphating, without successful maintenance of the temperature, within a reasonably close range. The modern industrial pyrometer, with its easily read scale and recording devices, should be of the greatest help in temperature control. Successful sulphating roasts have been made, in an experimental way, upon ores so low in sulphides (copper) that the amount of SO_2 in the interstitial gases must have been very small, so that the factor of temperature and, possibly, also of catalytic action of certain gangue minerals must have been particularly effective.

Temperature for sulphating roast for copper ore should be such (500 to 590°C.) that ferrous sulphate will be completely broken up and that cupric sulphate will be so far as possible undecomposed (580 to 700°C.).³ For the seven-hearth MacDougall-type furnace, with trapped pockets for passing ore from hearth to hearth (to prevent flashing), Mackay gives hearth temperatures as follows, first (top) to seventh, respectively: 420, 450, 490, 550, 590, 590, and 450°C. For low-grade ore (33 per cent zinc) at Great Falls, hearths one to seven, respectively, showed 440, 550, 605, 630, 650, 660, and 580°C.⁴ While the later studies of the sulphating roasting have had to do with low-grade semioxidized ores—of copper, for example, and of intimately mixed sulphides of copper, zinc, and lead, as in Coolbaugh process—earlier studies along this same line developed the well-known Ziervogel process for the recovery of silver by sulphate roasting of argentiferous matte, which found application in Germany in 1848 and at Argo, Colo., in the late seventies to the early nineties. Electrolytic refining has made such a process as the Ziervogel obsolete for mattes, but for mixed copper, zinc, lead sulphide, or semioxidized ores the sulphating roast may again find a field.

Chloridizing roasting and the recent "chloridation" of the volatilization process are identical, *i.e.*, the roasting of the prepared ore with a chloride, usually common salt, to convert the original mineralogical form of certain metals into chloride. Chloridizing roasting of silver ores was first introduced in Vienna by Born, and combined with the Cazo process in 1786.⁵ The early use of the Cazo process was as a preliminary to barrel amalgamation, but it found, temporarily, wide application also in the "hyposulphite" lixiviation process for silver ores. The chloridizing roast has persistently maintained a place in the literature of hydrometallurgy, particularly for copper, since the advent of the cyanide process for silver, but without important practical application, except possibly in the Longmaid-Henderson process for leaching

¹ *Eng. Mining Jour.-Press.*, Apr. 15, 1922, 629; U. S. patent 1376025.

² RALSTON, "Electrolytic Deposition and Hydrometallurgy of Zinc," 1921, p. 35.

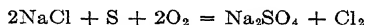
³ *Cf.* also Fig. 1, *Eng. Mining Jour.-Press.*, Apr. 15, 1922, p. 630.

⁴ *Chem. & Met. Eng.*, Feb. 9, 1921, p. 246.

⁵ *Bur. of Mines, Bull.* 211, p. 6.

pyrite cinder. Under the term "chloridation," it has quite recently appeared again as the first step of the proposed "volatilization process." But the facility with which chlorides of copper and other metals are volatilized, as shown by these studies, indicates that the chloridizing roast would undoubtedly have produced enormous metal losses in many of the processes for which the chloridizing roast has been recommended in technical literature as a preliminary step. In chloridizing blast roasting, in which the ore contains 5 to 10 per cent water, Keep maintains that there is "no volatilization loss."¹

The chemical reactions of chloridizing roasting (and "chloridation") are complex and not definitely known. In applying the chloridizing roast to preparation for "hyposulphite" leaching process, the reaction was generally assumed to be



In this process the general custom was to supply the sulphur in the form of pyrite, along with the salt.

Roasting Practice.—As roasting is generally a preliminary step in a major metallurgical process, it is necessary to choose a method appropriate to the conditions. For instance, if lead is the prime metal to be recovered, no copper whatever being present, it will be desirable to effect an almost complete elimination of the sulphur before smelting, thus avoiding the formation of an undue quantity of matte which locks up metal during retreatment. It is desirable also that the roasted product should be in lumpy form for the lead blast furnace. If copper is the metal under treatment, a complete elimination of sulphur may not be desired, but the degree of oxidation should be such that in the subsequent smelting a suitable grade copper matte shall be produced.

The mechanical character of the roasted product should be coarse if it is to be smelted in blast furnaces, and fine if it is to be smelted in reverberatories. In the latter case, it is desirable also to avoid the undue formation of magnetic oxide, as this is difficult to flux in the reverberatory whose reducing effect is slight. If zinc is to be recovered by retorting, a complete elimination of S ("dead roast") is necessary, involving the breaking up of the ZnSO_4 compounds which tend to form during roasting. The product should be reasonably fine in character (say 0.04 to 0.08 in.)² so as to permit of an intimate mixture with the pulverized carbon in making up the charge for the retorts. Thus there is a wide range of variation in requirements and conditions, and the choice of a roasting process should involve a consideration of the following factors: (1) Required mechanical condition of product in view of subsequent treatment. Thus, if the ore is to be leached, porosity should be preserved and the formation of insoluble compounds controlled; if the ore is to be smelted in blast furnaces, the product should be coarse and cellular; if in reverberatories, the product may be fine. (2) Required chemical condition of product in view of subsequent treatment, *e.g.*, oxidizing, reducing, sulphatizing, chloridizing, avoidance of formation of ferrites, etc. (3) The chemical composition of the original ore and associated minerals. (4) The mechanical condition of the ore as received (coarse or fine) and the question of crushing. (5) Elimination of undesirable elements and impurities, as S, As, Sb, etc. (6) Utilization of sulphur gases, requiring a gas rich in SO_2 ; also the question of smoke damage. (7) Control of temperature and air supply as in sulphating roasting. (8) Avoidance of losses of recoverable metals, especially the precious metals, by excessive temperatures; losses in dusting or by weathering and leaching (heaps and stalls).

¹ *Eng. Mining Jour.*, Feb. 6, 1915, p. 267.

² SCHENABEL, Vol. 2, p. 113, 1907ed.

The most desirable combination of operations is that one which yields the most satisfactory commercial result in the long run.

The furnace gases containing the eliminated S in the form of SO_2 or SO_3 may be wasted into the atmosphere, or may be utilized in the manufacture of sulphuric acid. The destructive effect of sulphur gases on vegetation is apt to become a serious question in thickly populated communities, and in many localities the percentage of SO_2 in the gases which smelters may turn out into the atmosphere is limited by law.

If the ore comes in lumps practically free from fines, the choice of heaps, stalls, or kilns may be indicated, though such practice is rare in these days.

If the ore is fine, it should be treated either on a reverberatory hearth or by blast roasting, and the advantages of these modern methods are so great that it usually pays to crush the coarse ores before roasting.

Essential Requirements.—The primary conditions essential to good roasting may be conveniently divided into three classes: chemical, thermal, and mechanical requirements.

Chemical Requirements.—As iron pyrites (FeS_2) is one of the most common of the natural metallic sulphides, it will be selected as typical. When FeS_2 is gradually heated to a low red heat with exclusion of air, one atom of S is expelled as elemental S, leaving a magnetic sulphide, FeS . In the presence of air, however, this reaction will be modified by the burning of both atoms of S, forming SO_2 , and leaving the iron in the ferric form, with perhaps some Fe_3O_4 , depending upon the temperature and air supply.

The other common sulphide minerals behave much the same as pyrites, except that they do not—with the exception of chalcopyrite ($\text{Cu}_2\text{S.FeS}_3$)—yield any sublimate of metallic sulphur, their ignition and incandescence temperatures are generally higher than for pyrites, and the resulting oxide degree will depend upon the ore that is being treated. The temperature at which the principal metallic sulphides will ignite differs for the different minerals and is affected also by the size of the particles. Table 1, on page 323, from Hofman,¹ gives the ignition temperatures for certain size grains. SO_2 gas in the presence of oxygen and in contact with hot Fe_2O_3 or SiO_2 tends to form SO_3 by catalysis. It is important to bear this fact in mind in considering reactions taking place in a roasting furnace, especially when questions of the formation and decomposition of sulphates are involved.

The progress of the roasting and the formation of sulphates of the metals are greatly influenced by the temperature and the presence of SO_2 and SO_3 in the furnace gases. Hence, a proper understanding of the behavior of the sulphates is important.² The conversion of SO_2 into SO_3 or the dissociation of SO_3 into SO_2 and O are reversible reactions and vary with the temperature and vapor pressure of the gases in the contacting atmosphere. The partial pressures of the ingredient gases are proportional to the relative volumes of these gases in the mixture, normal atmospheric pressure being taken at 760 mm. of mercury. In 100 volumes of air at normal pressure, 760 mm. mercury, there are 21 volumes of O and 79 volumes of N, giving partial tensions as follows:

$$21\%_{100} \times 760 = 159.6 \text{ mm. partial pressure of oxygen}$$

$$79\%_{100} \times 760 = 600.4 \text{ mm. partial pressure of nitrogen}$$

Now assuming that the interstitial gases immediately surrounding the ore particles have the analysis, by volume, as given in the following example, the partial pressures of the gases will be those here shown:

¹ HOFMAN, "General Metallurgy," p. 404.

² *Supra*, Sulphating Roast.

	VOLUME PER CENT	
SO ₂	10 per cent × 760 =	76 mm. partial pressure
SO ₃	4 per cent × 760 =	30 mm. partial pressure
O.....	18 per cent × 760 =	138 mm. partial pressure
N.....	68 per cent × 760 =	516 mm. partial pressure
	100 per cent	760 mm. total pressure

It is evident that the most active roasting atmosphere will be one in which the partial pressure of oxygen is a maximum. Nitrogen is an inert diluent: SO₂ and SO₃ play the double role of weakening the oxygen effect and even reversing the reaction and promoting the formation of sulphates. Good roasting, therefore, requires a removal of the sulphur gases from contact with the material, and a continual renewal of the oxygen supply. If, however, SO₂ is allowed to accumulate in the contacting atmosphere to the extent of 9 per cent by volume, the roasting effect will automatically cease.

The following reactions are typical;

1. $\begin{cases} 2\text{FeS}_2 + 11\text{O} = \text{Fe}_2\text{O}_3 + 4\text{SO}_2 \\ \text{Fe}_2\text{O}_3 + 3\text{SO}_3 \rightleftharpoons \text{Fe}_2(\text{SO}_4)_3 \end{cases}$
2. $\begin{cases} \text{Cu}_2\text{S} + 4\text{O} = 2\text{CuO} + \text{SO}_2 \\ 2\text{CuO} + 2\text{SO}_3 \rightleftharpoons 2\text{CuSO}_4 \end{cases}$
3. $\begin{cases} \text{ZnS} + 3\text{O} = \text{ZnO} + \text{SO}_2 \\ \text{ZnO} + \text{SO}_3 \rightleftharpoons \text{ZnSO}_4 \end{cases}$

For every vapor pressure of SO₃, there is a critical temperature below which sulphatization will be favored, and above which the sulphates will be dissociated, and revert to the oxide condition with expulsion of SO₃. Table 2 (page 646) gives these critical temperatures and pressures for certain important sulphates.

As the sulphates play a very important part in securing any desired result in a roasting operation, it will be clear from a study of the table (page 646) that a careful control of the two factors, temperature and vapor pressure of the contacting gases, is essential. Thus, in blende roasting, at a temperature of 720°C. the dissociation tension of SO₃ is 24 mm. This would correspond to a gas analysis of approximately as follows; SO₃, 3; O, 20.4; and N, 76.6 per cent. If the temperature remains constant at 720°C., and the partial pressure of SO₃ rises above 24 mm., or, in other words, if the analysis shows more than 3 per cent SO₃ in the contacting gas, the formation of ZnSO₄ will be promoted, while if the partial pressure of SO₃ drops below 24 mm., indicating less than 3 per cent SO₃ and correspondingly more free O in the contacting gases, the ZnSO₄ will tend to revert to ZnO. If blende is being roasted as a preliminary to leaching, the formation of soluble sulphates to a certain extent is desirable, while if a recovery of zinc by distillation is sought a rapid breaking up of the sulphates with as complete an elimination of S as possible is the end in view, and this means high temperature (short of melting) and a rapid removal of the sulphur gases. Thus, it will be evident from the table why it is more difficult to obtain a "dead roast" with ores of zinc than with those of other metals. In roasting blende on a rabbled hearth, a given particle will be repeatedly submerged in an intense interstitial sulphurous atmosphere and then exposed to the active influence of the oxidizing furnace gases. When the particle is exposed, the zinc sulphate tends to dissociate to oxide, and when submerged, to resulphatize. Accompanying these successive reversals of the reaction there is a gradual loss of sulphur, but the operation as ordinarily conducted requires a prolonged high temperature, and is slow.

In blast roasting, the fundamental conditions are less conducive to the formation of sulphates, and when once dissociated there is little chance for a resulphatization. This will be discussed further under the head, Blast Roasting Blende.

Arsenides and antimonides present difficulties in roasting because their higher oxides As_2O_5 and Sb_2O_5 tend to form very stable arsenates and antimonates with the various metallic oxides present. These objectionable compounds can be decomposed by reducing agencies, such as SO_2 gas, or by the admixture of carbonaceous matter in the roasting charge, which will produce the lower oxides As_2O_3 and Sb_2O_3 , which are readily volatile. A repetition of oxidation and reduction is necessary to drive out As and Sb effectively.

Thermal Requirements.—In Table 1 (page 323) are given temperatures at which certain sizes of particles of the various ores will ignite. To sustain active roasting, an adequate temperature must be maintained or the process will cease. If raised to the fusing point of sulphides, these will melt and thus stop the roasting. Certain free-burning sulphides (*e.g.*, iron pyrites) once ignited will roast without the aid of any external heat, provided excessive heat loss by radiation and otherwise is prevented.

Ordinary blende will self-roast on a rabbled hearth down to 8 to 10 per cent¹ without extraneous heat. Recent improvements in mechanically rabbled multiple-hearth furnaces, with careful conservation of heat by massive brick walls, and skilful by-passing of the gases and material between certain hearths, have succeeded in "dead roasting" blende without extraneous fuel. But roasting practice generally falls far short of the theoretical, and the roasting operation often has to be assisted by outside fuel both to ignite the material and to maintain the reaction. Especially is external heat necessary toward the end of the operation when the heat developed by the oxidation is waning and a high heat is needed to break up the sulphates and eliminate the residual sulphur. Such extra heat is furnished by fireboxes burning coal, oil, or gas and delivering their hot products of combustion over a reverberatory hearth upon which the ore particles are spread out in a thin layer. The heat is usually applied near a point where the roasted ore is discharged, and the temperatures are allowed to decrease gradually toward the feed end, as is proper for the best results. In general, the roasting temperature may be described as a bright red heat.

When the roasting is done in a reverberatory roasting furnace, a low roof is desirable, because the hot oxidizing gases which tend to hug the highest region of the arch are forced into close contact with the ore on the hearth: hence, it is considered good practice to design the arch as low as the space needed for the stratum of ore and the passage of the rabbling mechanism will permit.

Mechanical Requirements.—In addition to the chemical and heat requirements already mentioned, there are certain mechanical factors that must be provided for effective roasting, all of which are directed toward securing the maximum of reactive contact between the sulphide and the oxidizing gases.

This condition will be aided by the following factors: (1) Small granules, though not an impalpable powder. (2) A vigorous movement of the gases, which will constantly bring fresh oxygen into contact with the surface of the ore, and at the same time remove the spent gases, *i.e.*, the residual nitrogen and the sulphur gases. (3) Constant exposure of fresh sulphide surfaces to oxidation. In hearth roasting this is accomplished by frequent stirring, in blast roasting and heap roasting by causing the air to permeate evenly through the mass. Attention to these three primary requirements is essential to good roasting practice.

The size of grain usually specified as most desirable for lead ores where low sulphur is wanted is minus $\frac{1}{4}$ in., but $\frac{1}{2}$ in. may be small enough for copper ores where only partial elimination of S is wanted. Heavy dusting losses accompany the treatment of

¹ INGALLS, "Metallurgy of Zinc," p. 41.

very fine and dry material, in rabbled furnaces, in falling from hearth to hearth in multiple-hearth furnaces, or in subsequent handling. This dust loss is largely absent from blast-roasting practice, as will be explained in the appropriate section.

Roasting Methods.—Roasting may be performed in either one of four general classes of roaster: (1) Roast heaps and stalls (heap roasting); (2) reverberatory roasters; (a) hand rabbled (hand reverberatory), (b) mechanically stirred, of multifarious styles, (c) revolving cylinders (Brückners, etc.) (muffle roasters, kilns, hindered- and free-falling shaft furnaces (Gerstenhofer and Stetefeldt and the modern flash roasting) may be included as types of above classes); (3) blast roasting in pots, by the Huntington-Heberlein, Savelsberg, Carmichael-Bradford, McMurtry-Rogers methods; (4) blast roasting in thin layers (Dwight-Lloyd system): (a) intermittent down-draft pans, (b) continuous sintering machines.

A résumé of the efficiency factors of the above classes is:¹

	Efficiency, pounds per square feet per day	Character of product for blast furnace
1. Roast heaps and stalls.....	5 to 20	Good
2. Reverberatory roasters:		
a. Hand roaster.....	24 to 35	Fair
b. Mechanical average conditions.....	33 to 75	Too fine
Mechanical special conditions.....	150	Too fine
c. Revolving cylinders.....	128	Too fine
3. Blast roast in pots.....	500 to 900	Excellent
Average.....	600	Excellent
4. Blast roast in thin layers (Dwight-Lloyd system):		
a. Intermittent down-draft pans.....	1,000 to 2,000	Ideal
b. Continuous sintering.....	2,200 to 3,000	Ideal

Such a comparison of pounds per square foot of surface area per day can, of course, be taken only very generally, but the enormous difference between unit area efficiency of hand reverberatory, say 29, and the continuous sintering, say, 2600 lb. per sq. ft. hearth area per day, makes such comparison sufficiently discriminating.

Roasting in Heaps and Stalls.—Although now practically obsolete in large-scale operations in the United States, heap roasting may be indicated for isolated localities where more modern methods are not available. Hence this old practice is worth recording. Coarse ores, passing, say, a 2-in. ring, and from which fines have been removed, are adapted to heap and stall roasting. Where the proportion of pyrite in an ore is high, the 2 in. diameter may be considerably exceeded, but otherwise, and for mattes, the pieces should not much exceed this size. The ore must contain sufficient sulphur to sustain the combustion that has been started by a sublayer of fuel. The suggested method of an intermixture of fuel throughout the heap, to make up such deficiency, is hardly a feasible practice. The mineral in which the sulphur is present would determine very largely the lower percentage limit, which is somewhere

¹ DWIGHT, Efficiency of Ore Roasting, *Eng. Mining Jour.*, Dec. 30, 1911, p. 1267.

between 12 and 15 per cent S. On the other hand, where the sulphide content is high, the greatest care is necessary in regulating air supply that fusion of the sulphide may be prevented. To correct this tendency, a suitable admixture of lower grade sulphide may be made. At Cordoba, Peñarroya, and other places in Spain, cinders from boilers were added as diluent,¹ an example not likely to be followed.

To describe roast "heaps" as truncated "pyramids" is somewhat misleading, if the term "pyramid" suggests to the mind an equilateral base. The word "pyramid" itself, however, might well have been derived from primitive heaps, but it is much more likely to come from sacrificial or funeral pyres. The height of the heap is to some extent dependent upon the sulphur content (15 per cent S, 8 ft.; over 35 per cent

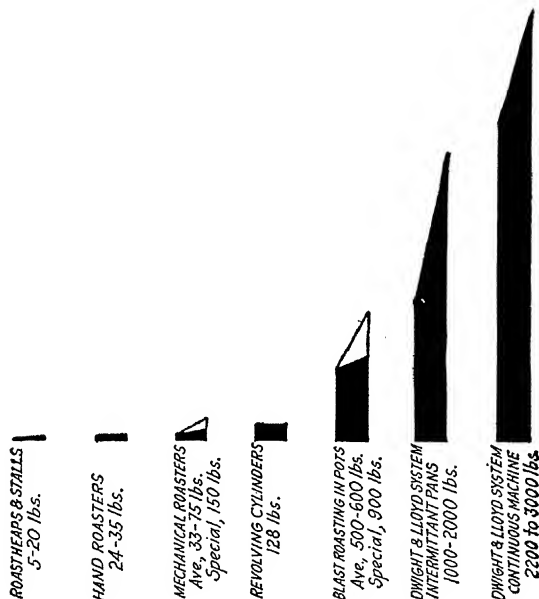


Fig. 1.—Pounds of ore roasted per square foot of hearth area per 24 hr.

S, only 5 ft.; average, 6 ft.).² Ability to control draft properly and unabated vigilance in this control are most important. Skill in successful regulation is attained by experienced observation.

The lateral dimensions of the heap depend upon various factors, such as manner of handling ore, whether it is received in railroad cars, Bleichert overhead rope tramway system, by steam shovel or simple barrows, the available areas of suitable level ground, etc. Generally, the larger the heap, the less the proportion of surface covering which must be retreated, so that good practice would favor long heaps, the width being determined by the method of handling. Hofman³ gives dimensions and tonnage of actual practice as follows: Stafford, Vt., 24 × 50 × 5 ft. high, 350 tons; Keswick, Calif., 20 × 500 × 8 ft. high, 15,000 tons; Mond Nickel Co., Ont., 40 × 80 × 15

¹ COLLINS, "Metallurgy of Lead."

² HOFMAN, "Metallurgy of Copper," p. 75.

³ HOFMAN, "Metallurgy of Copper," p. 76.

ft. high, 2500 tons; Canadian Copper Co., 55 × 100 × 10 or 12 ft. high, 1700 to 1900 tons.

The character of the ground is important. It should be hard, impervious, or clayey ground, fairly level, protected from surface drainage or wash, and free from depressions that collect water. A convenient arrangement is to have railroad tracks at suitable distances apart, upon a hillside, so that the raw ore may be unloaded from cars on an upper track, and the roasted ore into cars on a lower track, whether by barrows or mechanical device. This arrangement is especially convenient where sufficiently extensive area of level ground is unattainable. Of course, the uniformly level area of ground is less expensive to prepare, and, where steam shovel handling is possible, preferable.

The wood may be spread in a uniform layer over the entire area to be covered by the heap, either directly upon the ground or upon a uniformly spread layer of ragging or fines. If the quantity of wood is only sufficient to make a uniform layer of, say, 3 to 6 in., it would possibly be difficult to burn so thin a layer. The same quantity of wood, however, laid in parallel rows, at suitable distances apart, with occasional cross rows, would be preferable to too thin a uniform layer. At certain of the intersections of these lateral and cross rows, towers of wood and kindling should be carried to the surface of the heap. These towers and the horizontal rows of wood, when burned out, leave vents, or chimneys, and air passages that carry off the fumes and supply the air for roasting. Sheet-iron doors may be supplied for covering the entrances to the horizontal passages, or they may be regulated by coverings of ragging and fines. The passages are closed on the windward sides of the heap, especially during the earlier stages of the burning, or all passages may be closed for a time. Where there is a seasonal prevailing direction of wind, or draw of air, as up a valley, or canyon, considerable saving in the labor of placing the wood may be obtained by locating the heap so that the wood can be unloaded directly into a sort of windrow along what is to be the windward edge of the heap. When the wood is well ignited, the draft is checked by unloading ragging and fines along this edge, and the draft is regulated by removing and replacing, as may be required, the fines along the windward edge. Or, preferably, the wood may be placed along the two lengthwise edges. This method of placing the wood along the edges is applicable only in case of easily burning ore.

This crude method of roasting, although utilizing cheap, unskilled labor for handling the material, requires a most constant and skilled watchfulness during the long period of burning. It produces, however, an ideal product both physically and chemically (except that the sulphur is high) for the blast furnace, but it has a very low efficiency (5 lb. per sq. ft. of roast area),¹ involves the carrying a large stock of ore for a very small daily output, and necessitates enormous roast yards—960 sq. ft. to furnish 3 tons per day.² This area per ton-day is dependent upon the length of time required for burning and cooling a heap. This time varies from a few weeks, usually in smaller heaps, up to 3 to 4 months in large heaps. The large heap is more desirable because of the greater economy in the use of the roast yard space, and because of the much smaller proportion of exterior to be retreated and the usually better roast, if successfully handled. The cost of heap roast depends on so many factors that a stated average is of little value. Labor is the largest item: therefore efficient methods of mechanical handling and large tonnages make for low costs. Available data range from 15 to 20 cents per ton up to about a dollar. Metallurgical losses are also most variable. With copper ores and mattes, a high percentage of soluble copper sulphate may be formed in the heap, also sulphate of zinc and silver.

¹ DWIGHT, *Eng. Mining Jour.*, Dec. 30, 1911, p. 1267.

² HOFMAN, "Metallurgy of Copper," p. 74.

A collaborator has found that, in "a dead-burned zone" of a heap-roasted "first matte" assaying 5.7 per cent Cu, 51 per cent of the copper was water soluble. And in a stall roast, of an ore assaying 11.6 per cent Cu and 14 per cent Zn, 16 per cent of the copper and 22.8 per cent of the zinc were water-soluble. It is evident that such roasted material, if handled in the open, and upon flatcars, as is the customary manner of transporting, would, in rainy weather, probably entail heavy metal losses and, with copper, be destructive to rolling stock. There is practically no "dusting loss" during building the heaps, but heap-roasted ores, and particularly mattes, powder badly in tearing down the heap, so that in dry windy weather there is apt to be a considerable dust loss at this stage.

For the most part, however, heap roasting is prohibitive, owing to the lack of control of the destructive fumes which are fatal to all adjacent vegetation over very wide areas. Even in sparsely settled regions, the value of virgin forest that would be ruined by heap roasting upon any considerable scale has usually to be considered.

Stalls and kilns have not been extensively used in the United States, and their advantage over simple heaps in Europe seems to have been most largely due to the manner of handling the material and the ability to control the sulphur fumes. Stalls or kilns of brick construction are undesirable because the walls are so easily destroyed. If brick is laid in fire clay, there is not sufficient binding strength, and if in mortar, the lime is apt to become sulphated and swell and thus destroy the walls. The best success is with molded or cast-slag blocks. A number of American plants have in the past used roasting stalls with varying success (Eilers plant, Pueblo, Colo., East Helena plant, Mont., etc.). At Keswick, Calif., stalls were replaced by heaps.¹ Roasting stalls may well be considered as practically obsolete.

Hearth Roasting.—When the finely crushed ore is spread in a thin layer on a hearth of refractory material surmounted by a firebrick arch, under which hot oxidizing gases are caused to circulate in contact with the ore, the conditions are excellent for rapid and efficient roasting. All the factors necessary for good roasting, as previously pointed out, can be brought to bear, *viz.*, well-controlled temperature, vigorous air current, and frequent stirring or rabbling.

Hand-rabbed Furnace.—The simplest form of hearth roaster is the well-known hand reverberatory roaster, which was extensively used in lead-smelting practice in the United States up to 1905, when it began to decline and became obsolete in 1912, being superseded by the Huntington-Heberlein, and that in turn by the Dwight-Lloyd process.² Hearth, side walls, and roof are built of brick of refractory quality in the half of the furnace near the firebox. The arch should have as low a spring as is consistent with stability: hence, the better the quality of the arch bricks, the flatter it can be and the better roasting will result, as a high rise in the arch allows the hottest gases to keep at a distance from the ore spread out on the hearth. A rise of 10 in. in a span of 14 ft. is about the least that is safe. The frames of the working doors are of cast iron, provided with bars or rollers to support the heavy tool while rabbling, and are closed by a simple plate of sheet iron. Buckstays of heavy steel rails embedded in the ground and tied across the tops with steel rods and turnbuckles take the thrust of the arch and hold the door frames in place.

A furnace with a hearth 60 ft. long by 16 ft. wide will roast in 24 hr. about 15 tons of pyritiferous copper ore, reducing S from 35 to 7 per cent, or 31 lb. per sq. ft. per day, while, with ore that is not free burning, the output will be less, probably about 12 tons, or 25 lb. per sq. ft., per day. The depth of the ore bed is usually kept at 3 to 4 in. The ore is fed at the flue end of the furnace through hoppers in the roof, or shoveled through the last door, after the preceding charges have been successively

¹ HOFMAN, "Metallurgy of Copper," p. 82.

² HOFMAN, "Metallurgy of Lead," pp. 146, 147, 1918.

moved toward the firebox, following the drawing of the completed charge. The new charge, spread in a thin layer, soon dried, takes fire from the hot waste gases, and begins to roast. Every three hours a charge is drawn and the others moved forward one stage by long iron paddles (see description of tools below), with frequent rabblings and turnings over in between times. It is very arduous work, as may be seen from the description of the tools, and even a slave driver could not secure ideal results.

The tools for the old reverberatory hand roasters were the "paddle," the "peel," the "hoe," and the "rabble." The paddle had a blade 10×18 in. of $\frac{3}{8}$ -in. steel with a handle 12 ft. 6 in. long of $1\frac{1}{2}$ in. round iron, for half the length and $1\frac{1}{4}$ in. for the balance, with a crossbar at the outer end, 3 ft. long. The peel had a blade 7×20 in. of $\frac{5}{8}$ in. steel, the handle being about the same as the paddle, except that it had a rigid ring at the end, in place of the crossbar. The hoe had a blade 10×8 in. of $\frac{3}{8}$ - or $\frac{3}{4}$ -in. steel, the handle being 12 ft. long of $1\frac{1}{4}$ -in. round iron. The rabble had a

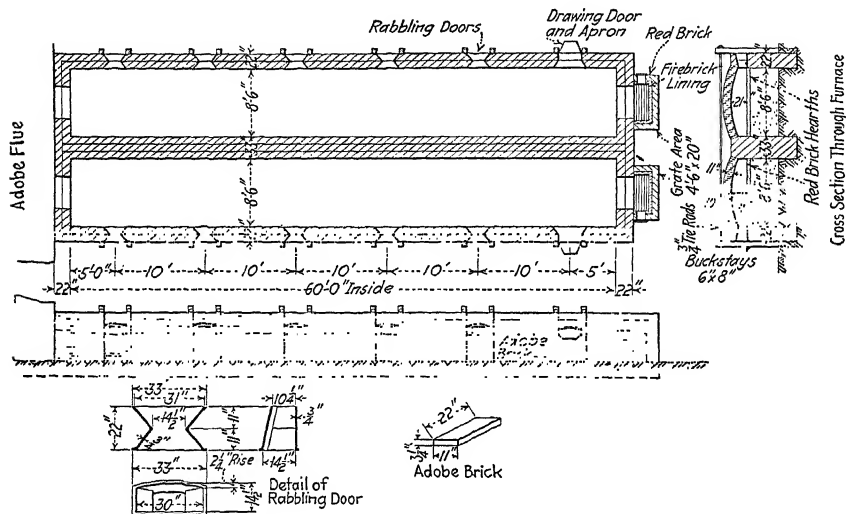


FIG. 2.—Adobe roasting furnace, San Luis Potosi, Mexico.

hoe-shaped blade, 6×8 in. of $\frac{5}{8}$ -in. steel and a 13-ft. handle of 1-in. round iron. The ore was moved forward by a sort of rowing-like action with the paddle. The peel was for breaking encrustations, the hoe for drawing the charge, and the rabble for stirring the charge and rabbling it by a back-and-forth action.

Two men every 8-hr. shift, or six men in 24 hr., would make up the crew, and 1 to 2 tons of long-flame bituminous coal per day would be about the fuel consumption. Flat grates and step grates might be used according to the nature of the coal. The cost of roasting usually varied from \$1.50 to \$3.50 per ton.

While this furnace has been almost entirely superseded in modern practice by more efficient apparatus, it has been described here because it still has its uses in special cases, and, moreover, it is the prototype from which have sprung all the many types of mechanically rabbled furnaces to be mentioned hereafter.

Until the advent of blast roasting, the hand roaster remained the one reliable stand-by of the lead smelter in the United States, though for a decade earnest unsuccessful efforts were made to adapt mechanical rabbling to sticky lead ores.

The hand-rabbled reverberatory roaster affords a comparatively inexpensive furnace, requiring a minimum of brick and ironwork. For installation in the frontier or inaccessible localities, or even as was done most successfully at one of the important smelting centers of Mexico, hand reverberatory furnaces may be constructed of adobe, with timber buckstays (8×6 in.) and 1-in. round-iron tie rods; the firebox of brick, firebrick lined, 4 ft. 6 in. \times 20 in., with steel rails for buckstays at the firebox. These furnaces are 60 ft. long, with a longitudinal middle wall separating the furnace into two parallel hearths, each 8 ft. 6 in. in width, with six rabbling doors, of special design (see Fig. 2) in the outer walls, the inner or middle wall, 33 in. thick (*i.e.*, thickness of length plus width of adobe) being the inner skewback for the two parallel arches. The outer walls are 22 in. thick (the length of an adobe).¹

The adobe arches stood up remarkably well, although eventually replaced, almost entirely, by red brick arches, with firebrick at the firebox end. In reality, the failure of the adobe arches was largely due to the deterioration of the corrugated iron roof over the roasters and the consequent admission of rain, rather than deterioration caused by the actual roasting service. The arches—span, 8 ft. 6 in., spring, 10 in.—were constructed of adobes edgewise, *i.e.*, 22 in. thick. Apart from the moderate first cost of these adobe roasters, they possessed considerable advantage in the roasting, due to the great conservation of heat because of the slight radiation loss. Therefore, with the long flame of the wood fire (oak wood), in spite of the inefficient character of the rabbling due to lack of physical strength of the Mexican laborer, the roasting result compared not unfavorably with reverberatory hand-roaster results on similar ores in Colorado practice of that day. Such a double furnace roasted 10 metric tons of leady ores down to 4 per cent sulphur, using 3 metric tons of oak cord wood. (Efficiency about 20 lb. of ore roasted per square foot of hearth per 24 hr.)

In all reverberatory roasting, by whatever type of furnace, whether manually or mechanically rabbled, the complete efficiency of the rabbling, both as to its character and its frequency, is of prime importance, so that, no matter what other adjuncts to the reverberatory hearth furnace have been added, or suggested, chief consideration has always been given to rabbling. Dr. Peters points out,² "the entire energy and ingenuity of the inventor and users of automatic furnaces were focused upon this single point." He actually states that they "succeeded" in solving the problem.

It is interesting to note in this connection that the amount of oxygen, the most essential reagent in roasting, received only casual consideration. Very little thought was given to the systematic supply of air to the furnace gases. Even with the introduction of the air-cooled rabble arm in the mechanical roaster, it was apparently the cooling effect that was foremost in the mind of the designer rather than the admission of oxygen, and in the more recent designs of air-cooled arms the air circulates through the system without escaping into the roaster atmosphere.

Mechanically Rabbled Furnaces.—It is not intended in this chapter to describe in detail the many makes of mechanically rabbled roasting furnaces, which are varied enough to meet every special need. They can usually be ordered from responsible manufacturers, who will supply detailed drawings. It is considered better for present

¹ Mexican adobes are $11 \times 22 \times 4$ in. thick (about), of any suitably plastic clay, with which is mixed a small amount of stable manure. The beneficial effect of the stable manure does not seem to depend on the binding effect of the straw—the impression received from reading the glossaries of Bibles—but is apparently due to a colloidal effect resulting, possibly, from tannin in the manure. This fact is noted because manure from straw-bedded stabling would not ordinarily be found in frontier localities, but such manure is suitable for adobe making, even though there be no visible straw. Comparatively recent investigations of German pottery clays have shown the colloidal effect of tannin occurring naturally in clay deposits.

² *Mineral Ind.*, Vol. 17, p. 283, 1908.

purposes to treat the subject in a semihistorical way, tracing the course of development of the different types and illustrating a few of the more outstanding ones in a more or less diagrammatic way. This development, together with a clear understanding of the fundamental chemical, thermal, and mechanical requirements of good roasting, should give a broad view of the subject and permit the reader to decide how his special roasting problem should be met.

Straight-line Furnaces.—In American practice, the mechanically rabbled furnace has developed gradually and naturally from its prototype, the hand reverberatory. The first departure (about 1885) was the O'Hara furnace, in which the arduous labor of hand rabbling was replaced by iron rabbles dragged by chains along the straight hearth of a reverberatory. This type was improved by Allen, who carried the rabbles on wheeled carriages running on tracks laid on the floor of the hearth. As the iron chains as well as the rabble blades were susceptible to attack by the hot sulphides and gases, Brown improved the furnace by spanning the hearth with a rabble frame resting on carriages running on tracks in narrow corridors on each side of the hearth and partially separated therefrom by curtain walls, leaving continuous straight slots of sufficient width to permit the arm of the rabbling mechanism to pass freely. The running gear and traction ropes were thus given reasonable protection from the heat and gases. The Brown patents covering the slot idea were the subject of much litigation and were held by the courts to be very broad and basic, and for many years they dominated the field of mechanical furnace design. The Brown furnace was made in two principal styles, the "straight-line," which was the original and more logical design, and the "horseshoe," so named from its shape, which was mechanically illogical, besides being wasteful of ground space, but had a certain vogue when circular furnaces of the Pearce type were popular.

The Pearce furnace, although not a straight-line furnace, may well be mentioned here because it belongs to the same period of development of the art of mechanical roasting and, while embodying many of the details of design of the straight-line furnaces, it formed the American prototype of the circular furnaces and undoubtedly contributed an important share in shaping the practice toward the modern MacDougall multihearth furnace. The Pearce furnace has an annular hearth served by radial rabble arms carried by a central column and projecting into the furnace across the annular hearth through a horizontal slot in the inside wall, which is sealed by a moving sheet-iron shield. This furnace was often built with two or three tiers of hearths. Following the Brown, other types of straight-line furnaces were developed, presenting differences in mechanical detail.

The Ropp furnace is a long straight-line furnace with a longitudinal slot or opening along the medial line of the hearth through which projects a vertical column carrying the transverse cantilever rabble arm spanning the hearth, the column being supported by a four-wheel carriage traveling in a tunnel under the hearth. A plurality of rabble carriages are connected in series by wire ropes passing over large horizontal sheaves outside the furnace at both ends which moves the carriages on an endless circuit through the tunnel and back to the place of beginning on a parallel track outside the furnace. The Ropp furnace has the advantage of allowing the rabbles plenty of time to cool, but is wasteful of floor space and not economical of fuel compared with some other types.

The Wethey furnace embodies many excellent points of design and is admirable for its simplicity both in structure and in mechanism. The brickwork is built as panels in a structural steel frame, the skewbacks of the arches being I beams hung from horizontal transverse girders. The carriages carrying the rabble arms run on tracks in the open air, and the slot is automatically opened and closed as the carriage in passing trips a series of overlapping swinging plates. The furnace is built with

single or multiple hearths and also in duplex form with the mechanism in an open space between the two blocks of hearths.

The Keller-Gaylord-Cole roasting furnace is also of the duplex type and had much vogue in Butte, Mont. It consists of two blocks each of six superposed hearths with the driving mechanism in the space between the blocks. The rabblers travel with a reciprocal motion, and when they are moving in an opposite direction to the travel of the ore the blades are lifted out of contact with the ore by a sort of "feathering" action. The rabblers are moved back and forth by tail ropes wound around a large drum actuated by a pinion and rack attached to a hydraulic piston. The slots are closed by metallic tapes which wind and unwind on spools as the rabblers move back and forth. This furnace made a good record in spite of its complicated mechanism.

An interesting type of the straight-line furnace is the Edwards furnace, developed in Australia and used to some extent in the United States. One design has a straight hearth 76 ft. long by $6\frac{1}{2}$ ft. wide with an inclination from feed to discharge end of $\frac{1}{2}$ in. per ft. One fireplace is at the discharge end and one on the side about one-third of the distance toward the feed. The rabbling is accomplished by 21 revolving horizontal arms carrying rabble teeth and making 1.5 to 2 r.p.m. The arms are attached to vertical spindles passing through the roof and held in position by iron frames supported on the side walls. The spindles are rotated by bevel gears, which are driven by a single line shaft running the entire length of the furnace. Each rabble sweep rotates in the reverse direction to its immediate neighbors, their circles of rotation overlapping each other almost to the centers. Thus the rabble arms, throwing the ore outward and inward, pass it on from one to another, causing it to take a zigzag course down the entire length of the hearth. There are dead spaces in the triangles between the intersecting circles of revolution and the side walls of the furnace. Another model of this furnace permits the whole structure to be tilted to a greater or less angle and thus increases the capacity under special conditions.

The Merton furnace has several superposed straight-line hearths, using the same general mechanism for rabbling as the Edwards. A still more recent development of Australian roasting practice is the Leggo furnace, which uses the same principle of rabbling sweeps revolving in overlapping circles. The Leggo furnace is made in multiple-hearth straight-line type, and also in turret form with four or six sweeps on each of the circular hearths. The Edwards, Merton, and Leggo furnaces claim great advantage from the light weight in the moving parts, ease in replacing rabble arms, and improved roasting, due to the low arches which this type of rabble arm permits.

The Zellweger furnace has found some use in roasting zinc ores. It is a straight-line furnace with a rolling rabbling mechanism attached to an axle connecting two large gear wheels which travel in corridors on either side of the hearth. The principal objection to this type is the high arch necessitated by the diameter of the rolling mechanism, which makes it very wasteful of heat.

Muffle Furnaces.—Muffle furnaces are particularly adapted to oxidizing roasts of sulphide ores where the S does not generate enough heat to decompose the sulphate formed in the roast and where it is essential that the roast gases be not contaminated with products of combustion of carbonaceous fuel.¹

For this purpose, the ore charge during all or part of the roasting period is contained in a chamber externally heated, so that the gases resulting from the roasting can be kept separate from the heating gases. Muffle furnaces were first introduced by Liebig and Eichhorn by arranging heating flues in pyrite burners of the Malet-Schaffner system, and then with success in zinc smelters by Hasenclever.²

¹ HOFMAN, "General Metallurgy," p. 426.

² HALL and HAYWARD, "Metallurgy," p. 233.

In the United States the idea was introduced by Mathiesen and Hegeler in 1881, but was in common use in Belgium and Germany before that. The Hegeler furnace is of the same type as the Spence; the Fahlding of the Edwards and Merton type; the Haas and Wedge are of the MacDougall type. The muffle furnace is expensive to build, to keep in repair, and to operate. "Every possible objection has been urged against the Hegeler furnace, among which may be mentioned these: It is difficult to regulate and expensive to build; it takes a comparatively large amount of room and is awkward in shape; it is difficult to arrange for easy and cheap handling of ore; it gives a fluctuating SO_2 gas and its fuel consumption is high. Regardless of all these faults, if it is properly built, this furnace can in the hands of trained men be operated to produce a well roasted product at a low operating and maintenance cost."¹

The Hegeler furnace as generally used in the United States has two compartments about 75 ft. long. Each compartment has seven hearths, the lowest two being of muffle construction. The rabble arms are dragged by chains or rods. The rake enters at one end and leaves at the opposite, and after being cooled by air enters the furnace again. Each shelf is closed at its two extremities by iron doors, with a mechanism for handling and drawing the rabbles. The shelves, or roasting chambers, are relatively high, about 2 ft., which tends to give poor gas, *viz.*, 3 to 4½ per cent SO_2 . The charge is stirred at intervals of 1½ to 2 hr. The capacity of the Hegeler furnace is about 40 tons per 24 hr. Fuel averages 20 to 25 per cent, power about 40 hp.²

Gas flues have more recently been in parallel instead of zigzag. The usual fuel is producer or natural gas. The Hegeler furnace is used in practically all zinc plants in the United States in which the sulphur gases are utilized for acid making. The improved Spirlet has had some use as at Kansas City and is said to produce richer gas.

Circular Furnaces.—The first circular roasting furnace is supposed to be the Parkes furnace, which was patented in England in 1850 and used for roasting copper ores. It embodied practically all the principles and many of the important details of the most successful mechanical roasters of today, *viz.*, those of the so-called MacDougall type. It had two superposed hearths, 12 ft. in diameter, a central vertical air-cooled shaft actuated by a bevel gear at the bottom, and drop openings between the hearths. It is said to have yielded efficient results, but the brickwork was constructed with insufficient reinforcement and racked itself to pieces rapidly, and so caused its abandonment. Thus, owing to a slight fault of design, the development of this most promising type of furnace was postponed several decades.

In 1873, MacDougall in England designed a modification of the Parkes furnace, which, by enclosing the brick structure in a 6 ft. diameter cylindrical iron shell, corrected the structural difficulty that had wrecked the Parkes furnace, and at the same time permitted the number of hearths to be increased from two to six. The ore was made to travel from the center to the circumference of one hearth and from circumference to center on the one below, and so on. The arches over one hearth formed the floor of the hearth above, thus greatly promoting the heat conservation. The weak point in the design was the construction of the rabble arms, which were rigidly attached to the center shaft so that the furnace had to be shut down and cooled off when it was necessary to make repairs to the arms or the rabble blades. Hence the repair item was so great that this furnace also had to be abandoned.

The idea of the circular furnace lay dormant for another two decades, until the urgent demand of American metallurgists for more efficient roasting methods caused the MacDougall principle to be resuscitated. The next improvement was made in

¹ CHASE, *Eng. Mining Jour.*, Oct. 20, 1917, p. 701.

² DE LUMMEN, *Eng. Mining Jour.*, June 10, 1916, p. 1023.

America by Herreshoff, who, in adapting the furnace to the roasting of fine pyrites ores for sulphuric acid manufacture, enlarged it and improved the construction in many respects, substituting for the fixed rabble arms of the earlier design air-cooled arms attached to the hollow central shaft by a simple locking device permitting the arms to be replaced without cooling off the furnace. This furnace was originally about 8 ft. in diameter, but has subsequently been built in much larger sizes and improved greatly in mechanical details, so that it is now one of the standard furnaces of this type, as will be described below.

Meantime copper-smelting practice in the western United States was finding that the straight-line roasting furnaces were expensive to operate, wasteful of floor space, and not good conservers of heat. Apart from the mechanical difficulties and other objections to the straight-line, horseshoe, annular, or other types of one-story reverberatory, the multiple-hearth furnace, for purely oxidizing roast, has, even with exactly equivalent rabbling, both as to manner and frequency, an advantage over the one-story furnace, due to the drop or fall of the ore from hearth to hearth, "inasmuch as probably 40 to 50 per cent of the sulphur oxidation takes place as the ore drops."¹ Terhune found, while experimenting with a Stetefeldt furnace, upon a leady matte containing 22 per cent S, that "its sulphur was half oxidized in the two seconds of time of descent."² This drop, which is advantageous for a purely oxidizing roast, becomes objectionable in roasting feriferous copper and zinc ores for *dilute* acid leaching, because of the formation of ferrite due to the high temperature of "flash" in the fall.

In 1892, Evans and Klepetko brought out a MacDougall furnace with numerous improvements, including water-cooled rabble arms, which permitted the diameter of the hearths to be considerably increased. With a diameter of 16 ft., six hearths, and a minimum of interruption for repairs, this furnace made a great record and set a new standard for roasting practice, especially in the field of copper smelting.

The Evans-Klepetko roaster, by its early adoption at some of the more prominent plants, became the most important of a number of variations of the MacDougall type of roaster. It represents an era when the furnaces in use were the result of modifications and improvements introduced by the metallurgists in charge of the smelting plants, in their efforts to meet their own necessities and without any particular thought of following up their introduction commercially. At the present time, the roasting furnace may well be considered in the light of a proprietary roasting machine, various "makes" being marketed in competition, such, for example, as the Herreshoff, the Wedge, and the Skinner, all of which are variations of the original MacDougall idea. The metallurgist, although thus so largely relieved of the responsibility for the constructive design, has, none the less, his full share of metallurgical and mechanical responsibility in adaptation of the furnace to his particular requirements, as determined by fundamental principles, both metallurgical and mechanical.

The Herreshoff furnace has a cast-iron, sectional, air-cooled, central driving shaft, with hollow air-cooled rabble arms, and with the MacDougall type of cylindrical steel outer casing. The sections of the central shaft are tongue-and-grooved, held together by bolts, the heads and nuts being protected by refractory material. The rabble arms are similarly bolted to the central driving shaft, air under slight pressure being forced to the ends of the arms and returning to the central shaft to be wasted or used as heated air for roasting. The blades slide transversely onto cast-iron plates, which, in turn, slide longitudinally onto the rabble arms. Arms and blades are removable through the hearth doors in the outer shell by stopping the arm opposite the door. Figure 3 shows a modern 12-hearth furnace of this type.

¹ *Eng. Mining Jour.*, Sept. 24, 1921, p. 491.

² *Trans. A.I.M.E.*, Vol. 16, p. 22, 1887-1888.

The Wedge roasting furnace is distinguished by its central air-cooled driving man-way or shaft ($\frac{1}{2}$ in. sheet steel, brick insulated) of a diameter sufficient (4 ft.) to permit replacement of the rabble arms and blades through the shaft. The rabble arms are water cooled, the supply piping also being accessible within the central shaft. The rabble arms are provided with cast-steel breech blocks of such design that the arms may be quickly removed and replaced and water connections made, without cooling

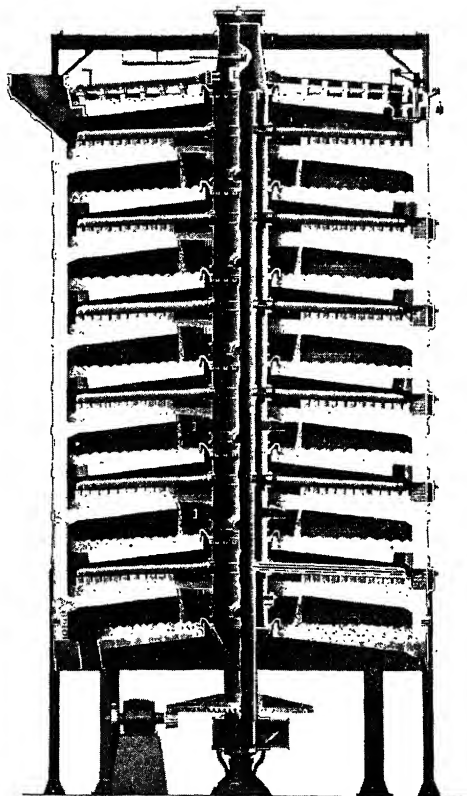


FIG. 3.—Herreshoff furnace.

or seriously delaying the furnace operations. The characteristic cylindrical sheet-steel shell ($\frac{1}{2}$ -in. plate) encloses the furnace. The skewbacks and various hearth bricks are of special pattern, the wall lining of the shell of 9-in. (generally) red brick.

In the Skinner roaster, the cylindrical steel-plate shell, which has been characteristic of the MacDougall type of furnace and has given the rigidity of construction which has been a feature of that type, has been abandoned for what might be termed a "cage construction." A series of 8-in. steel bands, at the skewbacks, of such stiffness

that the firebrick would be crushed before bending out of round, are supported by 18 large vertical channel columns. The entire frame is accurately centered and plumbed before brickwork is started. The hearth levels are accurately determined by the position of the skewback bands. The brickwork (the walls) between the skewback bands may at any time be readily knocked out and replaced for necessary hearth repairs, or the amount of radiation may be varied locally as the character of ore and requirements of roast may demand by using brickwork of varying thickness in the panels, or even sheet iron. The central actuating shaft, 30-in. diameter, is cast in sections, each section, except the top and bottom, holding rabble arms for two hearths. The rabble arms are fastened by a simple interlocking device by turning the rabble 90 deg.; no bolts are required. The rabble arms are air cooled, a vertical partition dividing the vertical shaft into two separate compartments, vertical partitions in the rabble arms articulating with the central vertical partition, causing the cooling air to pass up one side of the central column, through the rabble arms to the outer ends and back to the other compartment of the central column, thence to waste or for use as desired. The central shaft is supported on heavy conical roller bearings, and additionally aligned by four roller bearings at the top. Roller bearings are adjustable by set screws. Rabblers are removable from outside through suitably located hearth doors.

Review of Mechanical-roaster Types.—Since increased efficiency by frequency of rabbling has been the line of improvement, particularly in the oxidizing roast, the development of the mechanical roaster has naturally centered largely in this mechanical feature. On the other hand, in roasting for acid making or as a preliminary to hydrometallurgical treatment, the line of improvement has been to control the character of the gaseous products in the furnace atmosphere, the drawing of SO_2 -laden gases from the earlier hearths to finishing hearths, etc., or the by-passing of the ore charge itself in its descent from hearth to hearth.

In the development in the mechanism of rabbling in the straight-line and horse-shoe types of reverberatory roaster, the following features should be noted: chains moving in the roaster atmosphere—the O'Hara; cables moving in partially protected compartments or entirely outside—Brown, Wethey, Keller, etc.; carriage actuated through slot in hearth, like a "cable car"—Ropp; or by reciprocating movements in which rakes are provided with trippers so as to act upon the ore in one direction only—Spence, Hegeler (for acid essentially), Keller. But for ore roasting these reciprocating and vehicular rabbling mechanisms, of which there were such a number, have become practically obsolete. There are also furnaces with longitudinal one-story hearths and a series of rabble arms sweeping in circular, slightly overlapping fields, actuated by vertical rotating shafts, like the Edwards, Ridge, etc.

The Godfrey is a circular furnace with a horizontally rotating hearth, with stationary rabble arms. The Spirlet has alternately movable and stationary hearths, the rabblers for one hearth being attached to the lower side of the hearth immediately above. The Pearce furnace, with its vertical central shaft and attached radial rabble arms operating through a slot in the inner wall, is, in a very true sense, the link in development between the straight-line reverberatory and the multiple-hearth MacDougall type. Pearce rabble arms were air-cooled, air being forced through the air pipe in the central shaft and thence into the radial rabble arms, issuing through small holes, one between each two blades. The cooling air thus supplied additional air for roasting. Wedge, in increasing the diameter of the central shaft (to 4 ft.) sufficiently to permit the removal and replacing of the rabble arms through the central shaft, applied the Pearce furnace "well" to the MacDougall-type roaster. The most cursory glance at the variety of mechanical rabbling devices that were proposed or used for the straight-line reverberatory roasters makes most evident the advantage

out of contact with the walls of the combustion chamber until the reaction is complete and the particles have become partially chilled. The temperature of the gases leaving the combustion chamber near the top is about 1000 to 1050°C.

General practice is to pass the hot gases from the top side of the combustion chamber through a waste-heat boiler where the temperature is reduced from 1000°C. to about 350°C. The formation of SO₂ is reduced to a minimum by the high temperature maintained in the combustion chamber followed by the rapid cooling in the waste-heat boiler.

In producing SO₂ for sulphite paper-mill use, the gases are then scrubbed and cooled in a countercurrent scrubbing tower, usually packed with spiral tiles. The volume of the scrubbing water is controlled so that the temperature of the effluent water is about 80°C. at which point the solubility of SO₂ is extremely low.

Gas strengths of 10 to 12 per cent SO₂ are normal with recirculation of SO₂ gas. In most of the Australian installations, the gas from the flash roaster is partially cleaned of dust by passing direct from the combustion chamber to cyclone dust collectors and thence to the gas-cleaning system of the acid plant.

Trail System of Flash Roasting.—This system of flash roasting, developed by the Consolidated Mining & Smelting Co. of Canada, Ltd. of Trail, B.C., is applied to the roasting of zinc-sulphide flotation concentrates in connection with the company's large contact sulphuric-acid plants at Trail.¹

A modification of multiple-hearth furnace is employed, the central shaft and rabbling mechanism being used at the top for drying the concentrates and at the bottom of the furnace for finishing the roast, and in some cases for increasing the degree of sulphation when this is desired.

Wet blende concentrates are fed to the open top hearth and are rabbled inwardly and discharge upon the first internal hearth where they are dried by hot gases passing over the hearth. The dried concentrates are discharged at the periphery of the first internal hearth through a chute extending out from the wall to a screen or air classifier, the oversize being ground in a ball mill; the undersize and ground oversize are then elevated to a small feed bin from which the dry blende (85 per cent minus 200 mesh) is fed into the burner nozzle. This large nozzle is located just below the top internal hearth, and the blende is injected horizontally in a stream with all the air for combustion. The recycled cool gas and the gas used for drying the ore on the first internal hearth are sucked into this burner and mixed with the atmospheric air and air from the gas cooler.

It will be noted that the practice at Trail differs from the Nichols-Freeman practice by using a single downward current of air in the combustion chamber and withdrawing the waste gases at the bottom, instead of the countercurrents employed in the Nichols-Freeman system and the withdrawal of the waste gases at the top.

The fine flotation concentrates burn in suspension while falling vertically through a space about 20 ft. in height. They finally land on a lower hearth where rabble arms attached to the center shaft plow the partially roasted ore across this hearth to the center. Here the ore discharges upon the lowermost hearth where it is again rabbled toward the periphery from which the calcines discharge at a single point into the conveying apparatus for calcines. Dust collected from the gas cooler and Cottrell, when this is used, is normally returned by a screw conveyor through the side of the furnace just above the hearth forming the bottom of the combustion chamber.

A concentrate containing 30 per cent sulphur will flash roast down to about 8 per cent sulphur, and after the rabbling on the two lower hearths, the sulphur will be reduced to about 1½ to 2 per cent.

¹ U. S. patents 2030627, 2030628.

Units of this general type are in operation at the Grasselli Chemical Co. plant, Newcastle, Pa.; American Smelting & Refining Co., Corpus Christi, Tex.; New Jersey Zinc Company, Palmerton, N.J.; American Zinc, Lead & Smelting Co., East St. Louis, Ill.¹

The Brückner and Howell-White Revolving-cylinder Roasters.—The Brückner cylinder, for intermittent charges, and the Howell-White, for continuous feed, may be considered as, in a sense, movable-hearth roasters and are of simple and rigid construction. The Brückner and Howell-White found a prominent place in the chloridizing roast for such processes as the "hyposulphite" leaching and in early metallurgical practice—being revived to some extent in the earlier days of the present century—for roasting leady ores which would have proved difficult to handle in the then prevailing form of mechanically rabbled roaster. As there are no rabbles, the ore falling through the oxidizing field, this type, with its improved methods of firing and directing the flame against the falling ore, has much to recommend it. The advent of blast roasting undoubtedly hastened the abandonment of the revolving cylinder for metallurgical purposes, but the wide adoption of the revolving-cylinder type of furnace for burning cement clinker would seem strong testimony as to the mechanical desirability of the type. Excessive dust losses were particularly observable when using deflecting blades upon the sides of the cylinder, or a core, as in the Douglas revolving roaster, as at the Argentine, Kans., plant. A second fireplace was used in some cases in the Howell-White furnace practice, to roast the dust as it entered the dust flue.

The rotary kiln, similar to that used in the manufacture of cement, is being used to some extent for the roasting and nodulizing of flue dust, fine concentrate, etc. The kiln may be fired with powdered coal or oil, and the size of the sintered particle depends upon the amount of fuel used per ton of material. A 60-ft. kiln will treat 50 to 75 tons of material per day, depending upon its nature.

Modern copper-roasting practice calls for the delivery of the calcines to the reverberatory furnace as hot as possible, as a means of fuel economy. Special types of calcine cars have been developed to suit the conditions at the various plants, but all are based on the idea of heat conservation. The prevailing calcine temperature seems to be 500 to 900°F. Extraneous fuel, generally oil, is at times used on the lower hearths as a means of securing the desired calcine temperature, and also to increase the capacity of the roaster. Where the ores are lean in sulphur, the roasters are used simply as a means of drying the charge and preheating it for the reverberatories. At several plants where this is done, the oil used amounts to about 4 gal. per ton of feed.

Nodulizing has been experimented with as a competitor to briquetting for the treatment of flue dust. At Chrome, N.J., a 60 × 6 ft. cement kiln was used for this work, the operation being described as follows:²

The kiln, which had a 6-in. brick lining, was set at an inclination of $\frac{5}{8}$ in. to the foot and made 1 $\frac{1}{2}$ r.p.m. The kiln was fired with oil except when the fuel-oil market ran away, the top prices reaching nearly 6 cents per gallon; then a large firebox for coal, similar to those used in reverberatory practice, was tried, but did not operate satisfactorily, owing to the impossibility of keeping an even temperature at the mouth of the kiln when firing. Most cement kilns are now fired with pulverized coal, but the nature of the Chrome experiment did not permit of the considerable investment required for coal drying and pulverizing. Fuel oil is an ideal fuel for such work when its costs are low, and at the time these experiments were begun it was selling at about 2 $\frac{1}{2}$ cents per gallon. The oil consumption was lower than had been indicated by preliminary experiments with a smaller test kiln. With the 60 × 6 ft. kiln, a granular

¹ U.S. patents 1884348, 1963282, 2089603, 2120474, and 2120475.

² Addicks, *Trans. A.I.M.E.*, Vol. 49, p. 500.

sand could be made with about 8 gal. per ton, a first-class smelting nodule with 12 gal., and great chunks with 16 gal.

When perfectly clean, 75 tons of flue dust could be nodulized, but with a choked-up shell only about half this output was made. There is a decided tendency to form accretions within the shell, and it took some time for the operator to acquire the necessary skill to control the formation of the rings. Steady conditions of flame are necessary; if the kiln is overheated, semimolten material forms on the walls and a subsequent overchilling will plaster the sand on rapidly. Various mechanical contrivances for removing accretions were tried and abandoned; spraying with a hose was more effective; with reasonable care on the part of the attendants, and a few hours' work for two or three men once in two weeks, accretions were kept down to a point so that the cylinder delivered an average of 50 tons per day without difficulty.

The normal operating condition of the kiln is to have the first 40 ft. next the flue, acting simply as a preheater, without any real nodulizing action. The material then begins to stick and ride around, which is the first step toward the formation of nodules. The actual nodulizing should be completed in the next 15 ft., leaving the last 5 ft. for hardening to nodules in the cooler zone back of the flame. This is to keep the nodules from sticking together in a semifused mass when they drop into the receiving car. Nose rings will be confined to the 15-ft. nodulizing zone, where they are readily accessible from the end when the firing hood is rolled away. No trouble was experienced from flue dust blowing out of the kiln. It was expected that a considerable saving would result by delivering red-hot nodules to the blast furnace, but they smelted so much faster than the rest of the charge that better results were obtained by feeding the nodules cold. This probably would not be true in the case of nodulizing ores where the product was not self-fluxing. About 25 per cent of the flue dust fed would pass 100 mesh, and a screen test when making small nodules gave the following results: between $\frac{1}{2}$ and $\frac{1}{4}$ in., 54.21 per cent; between $\frac{1}{4}$ and $\frac{3}{16}$ in., 23.16 per cent; between $\frac{3}{16}$ and $\frac{1}{8}$ in., 20.6 per cent; between $\frac{1}{8}$ in. and 80 mesh, 1.9 per cent; through 80 mesh, 0.08 per cent. The oxidizing action, though marked in a small test kiln, was much less so in the larger kiln, nodules having 4.95 per cent S being obtained from flue dust containing 11.86 per cent S, or an elimination of 58 per cent.

While nodulizing is comparable with blast roasting, the two processes have different limitations, much as do blast-furnace and reverberatory smelting. Sintering requires a properly balanced internal fuel supply, while nodulizing is entirely independent of this. To nodulize properly, a material must have a sufficient interval between the temperature of becoming sticky (incipient fusion) and that of actual melting to give a practical range of operating flame temperatures. As a nodule is formed from individual grains stuck together, every piece has a solid structure and there are no defective ones full of unsmelted material. Mr. Addicks believes that the kiln is deserving of a wider application in the metallurgical field than it has as yet obtained, but the extent of this expansion can only be determined by experiment. While these notes cover but a single material, blast-furnace flue dust, and that from but a single plant, yet this material is much more like ore than pyrites cinder and other refractory products. It would be advisable to use a kiln 7 or 8 ft. in diameter to lessen the nosering difficulty, and this is in line with the experience in other classes of work. As to length of kiln for low-temperature work, 60 ft. is probably long enough; for refractory materials calling for a greater unit fuel consumption, longer kilns will pay in fuel saving. As far as operating cost goes, the crux of the whole matter lies in the cost of fuel. A single operator can attend to the firing of several kilns.

It may be mentioned in passing that experiments have indicated the nodulizing kiln to be well adapted to the agglomeration of flotation concentrates for subsequent blast-furnace smelting.

SINTERING BY BLAST ROASTING

Blast roasting may be defined as that method of roasting in which the oxidation effected by internal combustion is propagated within a mass of ore particles by air currents forced through the mass. It differs radically from the hearth-roasting methods previously described, wherein the oxidizing gases are in contact only with the surface of the mass, which, therefore, has to be constantly stirred in order to bring fresh ore particles to the surface of the mass and into contact with the oxidizing atmosphere.

The term "blast roasting" was suggested by the author in 1908 and promptly adopted by the industry at a time when the principal application of this procedure was in the treatment of sulphide lead ores in which the roasting was quite as much of a desideratum as the sintering; but as the important developments which quickly ensued included operations in which sinter was the desideratum with little or no roasting as in sintering iron oxide ores, the term came to have a broader meaning, and is still used in that broader sense as a matter of convenience.

Blast roasting requires more careful adjustment of chemical and mechanical conditions than hearth roasting, but when properly ordered it is capable of showing an efficiency per square foot of hearth area of three to twenty times as great as the best work in hearth roasting in the most modern type of mechanically rabbled furnaces. It can never entirely displace hearth roasting, however, because it is not always possible or expedient to provide the necessary conditions for successful blast roasting, and there are also other industrial factors that influence the choice of method.¹ Each method has its special sphere of usefulness. Sometimes they can with advantage be employed as successive steps to the desired end. Perhaps the most outstanding characteristics of the two methods lie in the character of their respective products. The product of hearth roasting is generally as fine and pulverulent as the ore was before roasting, while one of the principal advantages of blast roasting is that it may be made to turn out a coarsely agglomerated cellular product, commonly called "sinter," which is peculiarly favorable to blast-furnace smelting.

It does not follow, however, that merely forcing air through a mass of pulverized sulphide ore after igniting one surface will result in the production of "sinter." Depending on the conditions, any one of the following cases may be present: (1) A brief period of oxidation of the portion of the mass near the ignition, with a cumulative heat effect sufficient to melt together the adjacent sulphide particles to form an impervious layer of matted material that will prevent penetration of air currents and stop further roasting, or will form nodules of matte that resist further oxidation. (2) A complete effect of roasting, without sintering. (3) Roasting and sintering, more or less complete. (4) Sintering without roasting, as with oxidized iron ores, an admixture of fine carbon furnishing the fuel. (5) Calcination, or the expulsion of H_2O or CO_2 without further change, as in burning limestone; calcination, accompanied by sintering, as in the sintering of limonite or carbonate iron ores. All these phases, except the first (which is equivalent to failure), are used to produce certain desired effects in metallurgical work.

Numerous experiments are on record in which metallurgists have tried to increase the rapidity and efficiency of hearth roasting by the aid of jets of air, either impinging on the upper surface of the ore (as in the air jets in the rabbling mechanism of the Pearce turret furnace)² or in the porous-hearth furnace of J. E. Greenawalt,³ where

¹ See *Roasting Practice*, *supra*.

² HOFMAN, "Metallurgy of Copper," p. 132.

³ U.S. patents 839064, 904060.

the effect of rabbling was supplemented by a gentle draft of air passing upward or downward through the minute interstices of the hearth itself. Efforts in this direction, however, have generally been disappointing because they defeated their own ends. The high temperature produced by the more rapid oxidation of the sulphide caused premature melting of the charge and consequent interference with the normal progress of the roasting.

The great stumbling block in the way of increasing the efficiency of the mechanically rabbled hearth furnace has been that the amount of sulphur or other fuel that can be safely burned on a unit of area of hearth in a given time is distinctly limited. It is a function of the percentage of fuel in the ore (*e.g.*, sulphur) and the fusibility of the charge, the heat effect being augmented by mass action.

Technique of Blast Roasting.—Blast roasting became feasible when it was discovered how temporarily to hold in check and positively to control the mass action resulting from the rapid burning of the contained sulphur. The underlying principle of that control is to mix with the grains of raw sulphide a sufficient proportion of grains of an inert or refractory material or ore that will serve to isolate from one another the sulphide grains, holding each grain, as it were, in a little individual furnace with walls of refractory material. Moistening the mass also assists in the even control of the temperature by retarding the progress of the ignition through the mass, besides its other function of making the mass more permeable to air. Limestone and gypsum are specially effective diluents on account of the large amount of heat absorbed in their decomposition.

If, now, a stratum of slightly moistened material so constituted is ignited at one surface and a blast of air forced through it from the surface of ignition, the combustion will propagate itself through the mass in the direction of the air current. The sulphide particles in their little refractory cells, if held against agitation, will successively flash and shrivel up as the fierce oxidizing blast reaches them. For a given particle the combustion is almost instantaneous. The particle itself is transformed from sulphide into oxide, the heat of its combustion serving to dry and set fire to the next succeeding row of particles of sulphide. Thus, the combustion is propagated as a thin zone of intense combustion slowly moving through the mass in the direction of the air currents from the surface of original ignition to the opposite surface.

If the isolating particles of refractory material that have separated the sulphide particles during the oxidizing stage have a chemical composition such that they can combine with the metal oxides of the roasted particles to form a more or less fusible slag, the cumulative action of the heat developed by the mass effect in that immediate vicinity, intensified by the heat of chemical combination that is developed at the moment of nascency, will cause the oxidized particles and the refractory particles to melt together. Thus, for a brief moment there is a condition of fluidity; in other words, a thin film of liquid slag is momentarily present which is whipped and spun by the passing air currents into delicate gossamer-walled air passages and cells. If the particles are under too great pressure as from the weight of overlying material, these air passages will tend to close up while still plastic, and a more or less compact, slaglike mass will result. If, however, the pressure is light, as is the case where the layer of material is thin, the delicate cell structure will remain open, and a porous, cokelike mass will result, which has come to be known as "sinter" and which has been found to have many novel and favorable properties which modern metallurgy has not failed to utilize in many different fields. For this reason, thin layers are conducive to the production of the ideal quality of sinter.

There are other advantages of treating thin rather than thick layers in blast roasting which should be mentioned: (1) As the intensive thermal action is progressive, only a thin stratum is raised to the climax temperature at a given moment, and, con-

sequently, the bulk of the charge is either waiting to be reached, or has been finished and is waiting to be discharged. Hence, the smaller the ratio of the thickness of the ore layer to the thickness of the stratum of thermal intensity, the greater will be the efficiency factor. (2) Thin layers will give up their heat more readily by radiation or convection than thick layers, and there is less danger of premature melting by mass action. (3) Thin layers mean less supernatant pressure on the slagging particles, and consequently a more porous sinter.

One of the fundamental differences between hearth roasting and blast roasting is that in hearth roasting every particle in the charge is under treatment throughout the entire roasting operation, while in blast roasting the period of reaction for a given particle is extremely brief, not more than a minute or two.

This is proved by Hofman's temperature measurements,¹ which showed that, as the thin zone of reaction progressed through the mass, the temperature at a given point rose in 1 min. from under 100°C. to a sharp peak of 1000°C. or over, and then declined rapidly to 500°C. or less within 5 or 6 min. If, therefore, a layer of ore undergoing sintering with the correct proportion of internal fuel is dissected, a thin zone of intense heat will be found, accompanied by incipient fusion, moving with a smoldering action in the direction of the air currents with its plane bounded on one side by a coherent cake of rapidly cooling porous sinter, and on the other side by a stratum of moist, unroasted fine ore—awaiting its turn. The phenomena can best be compared with the gradual burning of a well-seasoned cigar, the tobacco corresponding to the sulphide ore, and the ash to the sinter (see Fig. 6). When there is an excess of internal fuel, the zone of reaction is less clearly defined, because of the slower cooling of the overheated mass.

Blast roasting requires external heat only for the primary ignition of the combustible elements at the surface of ignition, after which the combustion proceeds by a smoldering action within the mass supported only by the air currents. Any application of external heat is detrimental, as it interferes with the characteristic progressive reaction and causes overheating and premature matting of the sulphides and stoppages of the air passages by searing over the surfaces where the air enters the mass. Likewise, any stirring or rabbling of the charge is contrary to the theory of efficient blast roasting, because it disturbs the even transmission of the ignition effect through the mass and, by breaking up the close, brooding contact between the particles, which is necessary for the transmission of the ignition from the burning particle to its neighbors, tends to cause the temperature to fall below the degree necessary for maintaining the sintering and causes the fire to die out.

The air supply must be vigorous enough to maintain this internal combustion and develop sufficient heat to propagate the characteristic reaction that involves a momentary temperature of 800 to 1200°C., in a limited zone. Failing these temperatures, sintering will not take place, and the fire will die out. The rate of travel of the sintering zone through the mass from the surface initially ignited is $\frac{1}{4}$ to $\frac{1}{2}$ in. per min., when treating thin layers (*e.g.*, 4 in. thick) by the downdraft method under ordinary operating conditions. This rate increases with the volume of air, but the volume, in turn, is limited by the fusibility of the raw charge and the difficulty of holding very fine material on the grates.

Thus, there is a sharp distinction in principle between hearth roasting, which during almost the entire operation requires external heat (whether it is furnished by furnace gases or by hot furnace walls), and blast roasting, which requires external heat only for the primary ignition.

Evolution of Blast Roasting.—In 1896 at a smelting works at Pertusola, Italy, two metallurgists, Thomas Huntington and Ferdinand Heberlein, after many fruitless

¹ HOFMAN, *Trans. A.I.M.E.*, Vol. 38, p. 126, 1907.

experiments in roasting galena ores by an air blast, found that premature melting of the sulphide could be prevented by borrowing from the old Flintshire lead process the expedient of mixing a quantity of burnt lime with the charge. They were amazed at the vigor, rapidity, and completeness of the reaction that ensued. It seemed all out of proportion to what might be expected from a mere air blast. From what is now known, the cause was simple, and based on the establishing of favorable mechanical conditions as above described, but these investigators surmised that it must be the lime that was performing some chemical function previously unsuspected, and they finally concluded that the CaO became CaO_2 and served as a vehicle for conveying nascent oxygen to the lead sulphide. So sure were they of this fact that they based their first patents in all the important countries of the world on the use of burnt lime for this purpose, thus relying on a reaction that careful research subsequently demonstrated to be fallacious. Their results, however, received wide attention, and the process was extensively adopted.

In Australia, Carmichael and Bradford, while conducting the H&H process, discovered that CaSO_4 was an intermediate product of the reactions and that gypsum was as good, if not better, a diluent as burnt lime; so they proceeded to take out patents for the use of CaSO_4 .

Next A. Savelsberg, of Aachen, Germany, reasoned that CaO in limestone ought to answer as well as either of the foregoing, which he found to be the case, and so he took out still a third patent, claiming limestone as a diluent. In the H&H process the blast roasting was usually proceeded by a preroast in a reverberatory hearth furnace (Godfrey furnace). The Carmichael-Bradford and the Savelsberg processes could treat a higher percentage of sulphur than the H&H without preroasting because of the great heat-absorbing properties of the gypsum or limestone which they employed as diluent.

All these investigators firmly believed that lime was endowed with some hitherto unsuspected chemical potency as a carrier of nascent oxygen and that the remarkable speed of the roasting was due to the extraordinary heat thus developed. Much scientific research was undertaken to try to account for the phenomena on this theory, when, as a matter of fact, there was no more heat generated than could be accounted for by well-known chemical reactions. Dwight and Lloyd demonstrated that the vigor of the reaction resulted from the advantageous manner in which the fuel element was brought in contact with the air currents, a purely mechanical matter, and that the real problem was not so much how to augment the evolution of heat, as how to control and dissipate it, as has been already explained.

Although Huntington and Heberlein were not at first aware of the whys and wherefores, they did succeed in working out a practical solution of an important metallurgical improvement. They are entitled to great credit as the pioneers in this new art, which has had such far-reaching developments. They made a great advance in the metallurgy of lead, by furnishing a method of desulphurizing galena that was three to five times as efficient as the best mechanically rabbled furnaces (if, indeed, these could have been adapted to roasting high-percentage lead ores satisfactorily) and fifteen to twenty-five times as efficient as the hand-rabbed reverberatory which had remained the standard device in the lead-smelting industry up to that time.

In these three variations of blast roasting, the operation was performed in large cast-iron pots with a grate in the bottom through which air under pressure was forced upward through the ore charge, the ignition being started by a layer of hot coals or red-hot ore placed on the grates before feeding the charge. The product was an agglomerated cake of desulphurized material partly porous and partly indurated, accompanied, usually, by a stratum of pulverulent, partially roasted but unsintered fines on the top. This procedure was commonly called "pot roasting" or "lime-roasting."

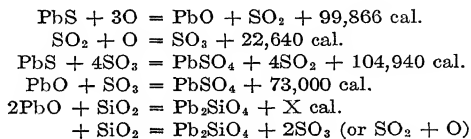
Other attempts at updraft blast roasting were made by J. W. Neill,¹ J. Williams, Rhodes and Klotz, H. Haas,² H. C. Bellinger,³ Cyrus Robinson, F. G. Kelley, and others. All these methods use an updraft and are intermittent in operation.

The next step in the improvement of blast-roasting methods was taken by Dwight and Lloyd, who discovered that, whereas it was impossible to sinter completely a thin layer of ore by updraft, on account of the agitation of the particles among themselves in the upper portion of the layer, caused by the air currents, it was possible to sinter the entire layer into a coherent cake by the simple mechanical expedient of holding against agitation the particles of the mass at the surface of exits of the gases. This could be accomplished by superposing a fine metal screen on the top surface of ore in an updraft pot; but the simplest way was to reverse the draft, drawing the air downward through the layer of ore and the grate upon which it rests, thus making the grate act in the dual role of support and restrainer. There was the additional advantage that by shifting the ignition to the upper surface it became possible to change the process from an intermittent to a continuous one. The utilization of this simple discovery has given rise to important economies in numerous established lines of metallurgical work and has furnished the metallurgist with a new product, "sinter," possessing peculiarly valuable characteristics, the use of which is continually opening up new possibilities of improvement in method in numerous fields of the metallurgical industry.⁴

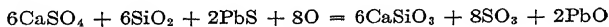
The Dwight-Lloyd process depends for its successful operation upon an application of the fundamental principles of blast roasting, already described. It is a mechanical improvement over the older blast-roasting process (1) through the adoption of the mechanical expedient above described for completely sintering a thin layer (2) by developing a continuous process. No special diluents or fluxes need be resorted to except as required to establish proper charge conditions, as elsewhere explained.

Chemistry of Blast Roasting.—The general principles previously enunciated governing good roasting apply to blast roasting as truly as to hearth roasting. While the differences are chiefly thermophysical, there has been so much speculation over the behavior of the galena in conjunction with various diluents that it may be useful to recite briefly the results of the best research.

Taking SiO_2 as a typical inert diluent, the reactions with galena may be expressed as follows:⁵



Richter's experiments established the fact that CaO and CaSO_4 do not appear to have any direct action on PbS and PbO . CaSO_4 , which may be the diluent, is decomposed by heat alone at 1200°C . and in the presence of SiO_2 starts decomposition at 1000°C . and is complete at 1250°C . The reaction



begins below 800°C . and is complete at 1000 to 1100°C .

¹ Hofman's paper on blast roasting and discussion, *Trans. A.I.M.E.*, Vol. 41.

² *Eng. Mining Jour.*, 1910, p. 1910.

³ U.S. patent 942052, Dec. 10, 1910.

⁴ U.S. patent 1020340, Dwight and Lloyd.

⁵ HOFMAN, "Metallurgy of Lead," p. 159, 1918.

Make-up of Charges.—To secure a satisfactory elimination of S and the production of a proper quality of sinter, certain conditions must be fulfilled. The heat developed by the oxidation must not be excessive, or premature melting will stop the roasting action. This tendency can be controlled by (1) preroasting the sulphides to remove excessive fuel value; (2) addition of diluents to the charge, sufficient to reduce the average percentage of fuel ingredients to a suitable extent. Heat-absorbing minerals, such as those containing CO_2 or combined H_2O , are most effective; (3) moistening the charge; (4) diluting the air with inert gas (not used). Huntington-Heberlein use (1) and (2); Savelsberg and Carmichael-Bradford use (2) and (3); Dwight-Lloyd use (1) or (2) to suit the special problem, and (3).

The limit of sulphur for direct blast roasting is generally taken to be about 16 per cent, and 12 per cent is better. Sulphur over 16 per cent makes too fiery a charge, with a tendency to matte formation and incomplete roasting. Where sulphur elimination is not important, however, as in some copper ores, charges containing as high as 30 per cent S have been sintered. Higher sulphur can be treated by Dwight-Lloyd methods than by the pot processes, because in the former there is greater facility in dissipating the heat and less tendency to mass action.

Preroasting for Blast Roasting.—The standard practice of the Huntington-Heberlein process employed the Godfrey furnace for preroasting. This is a circular furnace with a revolving hearth and stationary rabble blades projecting down through the roof. The hearth was usually 26 ft. in diameter, and the daily capacity was about 30 tons of material, reducing the sulphur from about 25 to 8 or 10 per cent. Other types of mechanical furnace were used for preroasting, such as the Ropp at Port Pirie, Australia; in fact, any good furnace is suitable.

Following the adoption of a Wedge for preroasting leady matte at Herculaneum, Mo., in 1911, in preparation for D & L sintering, that type of preroaster was extensively adopted in western plants.¹ The D & L apparatus itself can be used for preroasting under certain conditions, as will be described under the heading Double Roasting.

Whatever type of furnace is used for preroasting, great care must be exercised so to control the temperature that there shall be no premature fusion of the sulphides. This control is obtained by regulating the admission of air and the exit of furnace gases, by-passing them in some cases, and varying the rate of feed and the frequency of rabbling.

In this connection the peculiar behavior of the sulphur in certain sulphides should be mentioned when roasted by the downdraft method. When iron pyrites is present in the charge, there is a tendency for the extra atom of sulphur to distill off as elemental sulphur and condense in the interstices of the cold moist ore below, then melting and liquating downward in advance of the descending zone of high temperature until it is expelled through the grate either as drops of fluid sulphur or as sublimed flowers of sulphur. In updraft, any sulphur thus condensed tends to liquefy down into the fire and be expelled as SO_2 rather than as S.

Pot Roasting.—Space will not be taken to describe separately the Huntington-Heberlein, the Carmichael-Bradford, and the Savelsberg processes, since the procedure is practically identical, except for the preroasting, and even that difference was gradually obliterated. Although they are now almost obsolete in the United States, having given place to the cheaper and more effective D & L process, there are cases where the need for simple and impromptu apparatus makes a knowledge of their use desirable.

In addition to any preroasting equipment that may be needed, a pot-roasting plant consists essentially of one or more pots, blast apparatus capable of furnishing

¹ HOFMAN. "Metallurgy of Lead," p. 173, 1918.

1000 to 2000 cu. ft. of air per minute per pot, at a pressure of up to 15 to 20 oz.; a furnace or brazier to kindle hot coals for igniting the charges; adjustable hoods and stacks over the pot stands for carrying off the sulphur gases, which, being concentrated, are particularly irritating to the workmen; and suitable arrangements for making up the ore charges, mixing and delivering to the pots, and handling the product therefrom.

The pots originally used were small conical-shaped containers holding about a ton of charge. They were mounted on trucks and could be moved about at will. In American practice, the pots were of a general hemispherical shape, 8 to 10 ft. in diameter at the top, by 4 to 6 ft. deep, of cast iron, usually built in sections, and holding 8 to 10 tons of charge. The grate, also sectional, pierced by $\frac{3}{4}$ -in. holes, was about 6 ft. in diameter, set about 18 in. above the bottom of the pot. Under the grate there was a 5- to 6-in. air inlet, arranged so as to be quickly coupled to the blast nipple when the pot had been charged and placed in position for the blow. In the smaller plants, the pots were mounted on high supports and were tipped by means of a worm gear and hand wheel; in the larger plants, the battery of pots was served by an overhead traveling crane which could pick up a pot bodily, carry it to the Godfrey furnaces or elsewhere to receive the ore charge, replace it in the blowing stand, and, after the blow was finished, pick it up, pour off the unsintered fines on the top of the cake for retreatment, and discharge the agglomerated cake on the cooling floor, where it would later be broken up into fragments of suitable size for the blast furnace, either by men with sledges or by dropping weights on it.

In the operation, the grate generally is first covered by a bed of ashes, on which is spread a layer of burning coals or red-hot "primer" calcine from the preroaster, which will ignite the charge when the blast is turned on. On this the 6 to 9 tons of properly prepared charge are evenly spread, usually in a single batch, making a bed 4 to 5 ft. thick, but sometimes added as a number of small batches at intervals during the blowing period. The pot is now covered with the hood, and the blast is immediately turned on, at first under 4- to 8-oz. pressure, which is increased to 8 to 16 oz. after the charge is well ignited. The blowing continues for 8 to 12 hr., or until the roasting is completed to the top surface of the charge. At short intervals the pot tender examines the pot through doors in the hood and breaks down any channels or "blow-holes" that the blast may have formed through the bed, keeping the action as uniform through the pot as he can by poking or plastering the crater with fresh charge. Toward the end of the "blow," the blast pressure is again reduced, in order to decrease the agitation of the upper layers and reduce the amount of unsintered "fines" that always remains on the upper part of the charge, and which must be retreated. When the operation is completed, the pot is dumped and the cake broken up as described above.

The labor required to operate a pot plant is divided among charge crew, pot tenders, and crew breaking up cake. The actual number of men needed will depend

	Pot plant	D & L machines
Monthly tonnages treated.....	6,500 tons	10,000 tons
Men per 8-hr. shift.....	32	8
Tons roasted per man per day....	2.3	14.8
Cost per ton (average 1 year).....	\$1.54	\$0.50
Cost of plant, per ton-day capacity	\$300	\$146

on the size of plant, use of overhead crane, and other mechanical labor savers. At a number of plants formerly in use, the labor ratio varied from 2 tons of ore roasted per man per day to 8 tons per man per day, while the cost of pot roasting (*circa* 1910) was from \$1.50 per ton to \$5 or \$6, averaging perhaps \$2.50 per ton. The figures tabulated on page 307, taken from a plant which was at that time treating the same ores both in pots and on D & L machines, in approximately the same tonnages, illustrate one reason for the passing of blast roasting in pots.

Dwight-Lloyd Process of Sintering.—This process differs from the other methods of blast roasting in that the charge is treated in a thin uniform layer in such a manner that agitation of the individual ore particles is prevented, with the result that the entire charge is roasted and sintered to a porous cellular biscuit of proper size for subsequent blast-furnace treatment, thereby avoiding both the large proportion of unagglomerated fine particles from the top of the charge and the huge solid cake that result when blast roasting in pots without restraint against agitation at the top surface. In practice, quiescence of the ore particles is most conveniently obtained by drawing the air blast down through the charge, thus making the grate act as a screen to prevent agitation, but it may be accomplished in other ways.¹

The Sintering Machine.—Of the several types of continuous sintering machines that have been designed by Dwight-Lloyd, the straight-line machine is now the most widely used. The drum type² and the horizontal round-table type of machine³ are both limited to small capacities by mechanical considerations and have been entirely superseded by the straight-line machine in this country. In Europe a number of the round-table machines may be in use, especially in connection with the manufacture of sulphuric acid from the gases. Von Schlippenbach⁴ divided the wind box into sections, connected with separate fans, one of which draws off gases rich in SO₂ which are used for making acid, and the other withdrawing and wasting the lean gases from the other section. This modification may also be applied to the straight-line machine, and the many other advantages of the straight-line type over the round-table type have resulted in its use in the more important European installations.

Pans or shallow trays, with downdraft operated intermittently, have been occasionally used for sintering, as at Cerro de Pasco, Peru; Port Pirie, Australia; Midvale, Utah, and Anyox, B.C., but have usually been replaced by the continuous machine with its many advantages in operation and economy.

The D & L straight-line machine consists essentially of a heavy structural-steel frame that supports the suction box, feed mechanism, igniter, a pair of endless-track elements, the pallets or cars that carry the charge and the driving mechanism (see Fig. 5). The pallets are not connected to one another, but completely fill the track, making a closed circuit, except for a space of about 24 in. at the discharge end, and thus form a continuous conveyor for the charge. The ends and bottoms of the pallets are machined to assure tight-fitting joints both between adjacent pallets and between pallets and wind-box tops. At each end of the wind box is a machined dead plate, which seals against air leakage as the pallet approaches or leaves the wind box; and along the top edges of the wind boxes are adjustable soft-steel seal bars which make light contact with the inner side of the pallet, thus preventing air leakage at this point. In the larger machines this seal bar is now replaced by a flexible ribbon or band of special composition, which is held by suction in close contact with the side of the pallet.⁵ A pair of steel sprockets, driven by suitable gearing, engage four steel

¹ Cf. U.S. patents 882517, 916396, and 916397, A. S. Dwight.

² DWIGHT, *Eng. Mining Jour.*, Mar. 28, 1908, p. 649; *Mineral Ind.*, 1907, Vol. 16, p. 380; and U.S. patent 916394, A. S. Dwight and R. L. Lloyd.

³ HOFMAN, *Trans. A.I.M.E.*, 1910, Vol. 41, p. 756; U.S. patent 916392, A. S. Dwight.

⁴ U.S. patent, 1069191.

⁵ U.S. patent 1491778, by H. J. Stehli.

rollers on the pallet axles, to lift the pallets from the lower return track and push them forward along the horizontal upper track where the sintering takes place. Thus, while on this upper track, the machined joints between adjacent pallets are kept tightly sealed by the pressure of the following pallets, as the train of pallets is

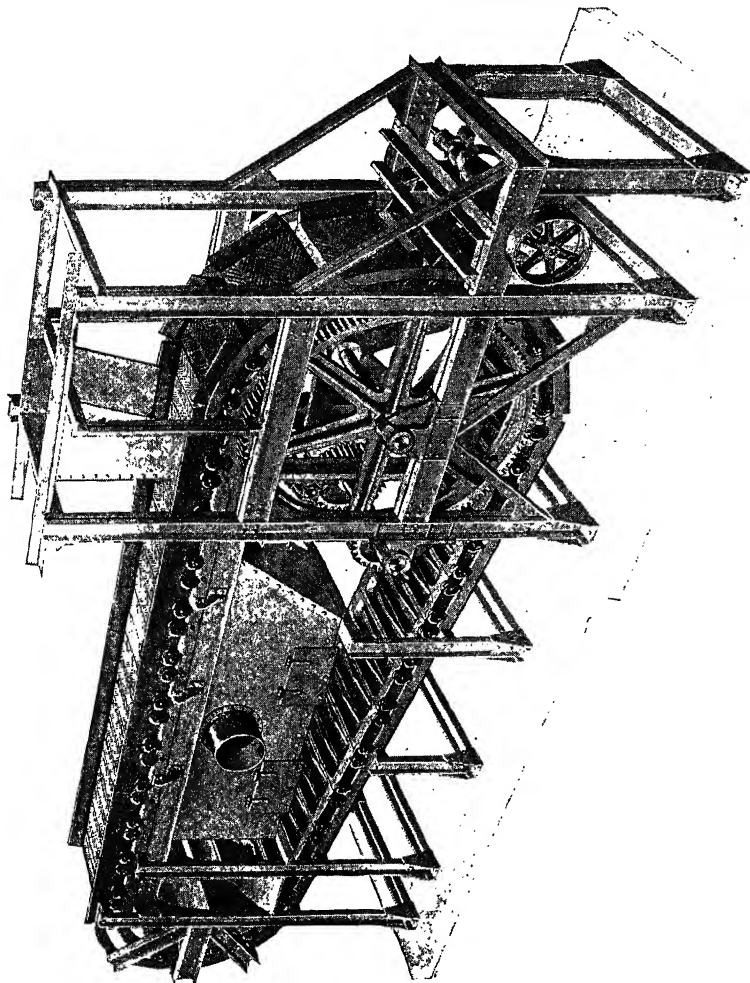


FIG. 1. t-Lloyd nigh ine si

pushed slowly along over the wind box. At the discharge end, the pallets are pushed off one by one over the brink formed by the curved guides where there is a space of 24 in. down which the pallet plunges to strike against the edge of the one ahead, the shock discharging the sinter cake and cleaning the grate-bar slots. The track is so

designed that, just before it drops over the end curve, the pallet is raised vertically half an inch, which breaks the cake loose from that on the following pallet. The pallets return down the lower inclined track by gravity, in the smaller machines, or are pushed back along a horizontal track by a second pair of sprocket wheels, in the large machines. Formerly, the section of outboard track beside the wind boxes was omitted, the pallets sliding along the top edge of the wind-box castings; but although this gave an excellent seal against air leakage it resulted in excessive wear on the contact surfaces, so the complete track circuit described above was adopted.¹ The outboard rails beside the wind boxes are set at a height such that the pallet bottom just clears the wind-box top, air leakage being prevented by the seal bars or flexible seal band above mentioned. Rate of travel of the pallet train is so adjusted that sintering is complete down to the grates as the pallet reaches and passes beyond the end of the wind box.

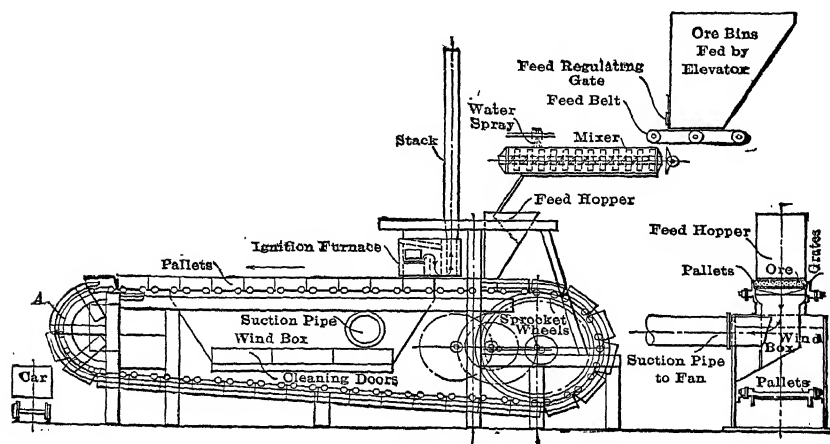


Fig. 5.—Diagrammatic sketch of Dwight-Lloyd straight-line sintering machine.

The wind box or suction box, of which there are one or more sections, depending on the length of the machine, is made up of castings and steel plate, or entirely of castings, depending on the service the machine is designed for. Cast iron resists the action of condensed sulphurous and sulphuric acids better than the steel plate, and also is more resistant to the abrasive action of such materials as iron flue dust, but a cast wind box is considerably more expensive and heavier. In general, unless temperatures are to be expected at which the sulphurous and sulphuric acids will be active or where iron flue dust is to be treated, the steel-plate wind box is preferred. The wind box is connected to a suitable high-vacuum centrifugal fan, which draws the air through the charge while it is passing over the wind box. The degree of vacuum depends upon the class of materials treated, varying from 4 to 16 oz. or even more, the higher vacuums being necessary with such tight charges as, for instance, iron flue dust sometimes makes. In general, the lowest vacuum that will draw through the charge the necessary quantity of air should be employed, as high vacuum has a tendency to cause the charge to pack and hence to aggravate the condition it is designed to overcome. The wind box is made of large volume, to act as a dust catcher to collect the small amount

¹ U.S. patent 1097592, by E. H. Laws.

of dust that may be drawn through the grates, and is provided with clean-out doors. Smaller wind boxes, with a separate dust catcher between the machine and the fan, may be used. With several of the larger size machines the wind-box elements have been reduced to mere nozzles connecting to a central bustle pipe, which is flushed with a stream of water to wash down the dust. Needless to say, this is the practice only where the gases are free from SO_2 .

Fans.—A special type of centrifugal fan has been developed to meet the exacting conditions found in sintering. The construction is particularly rugged throughout to withstand abrasion and corrosion, and repairs, when necessary, are simplified by making the casing sectional so that the impeller and shaft can be removed bodily. Large clean-out doors give access to the inside of the casting. The impeller is of the center-plate type with the shaft carried through the casing to large outboard bearings on each side, and the inlet is divided to admit half the gas to each side of the impeller in order to avoid unbalancing the latter. Water-cooled bearings lined with high-speed babbitt or ball bearings of ample size are used, and, where the gases to be handled are very hot, the shaft itself is water cooled.¹

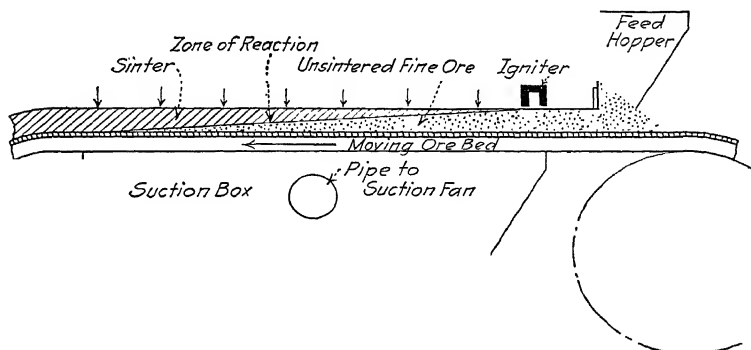


FIG. 6.—Longitudinal section through sinter bed.

The motors are direct connected to the fans by means of a rigid coupling and a cast-iron base plate carrying the motor and one of the fan bearings, thus ensuring positive alignment. A single sintering unit should invariably have its own individual fan, otherwise irregularities in operation will result.

Ignition.—The igniter, placed above the pallets at the forward end of the wind box, serves to set fire to the surface of the charge as it enters the suction zone. It may be fired with any available fuel—solid, liquid, or gaseous—and, of course, the type of the igniter itself will depend on the kind of fuel used. Distillate, powdered coal, or blast-furnace or by-product gas are most generally employed, as one of these is usually available and cheap. Lump coal, coke, charcoal, wood, gasoline, kerosene, producer gas, city gas, and other fuels have been used with entire success, but oil or gas is now most commonly used. The igniter itself consists essentially of a fire-box or burner of suitable type and a muffle or inverted trough that deflects the hot products of combustion uniformly across the surface of the charge on the pallets and protects the workmen and adjacent structure from the heat. It is placed at the front edge of the wind box, and above at right angles to the direction of motion of the pallets, so that the hot gases which are blown into the muffle at high pressure

¹ U.S. patent 1439491.

are drawn against and into the charge by the suction of the fan, and ignite the charge evenly across its exposed surface. As ignition requires only a few seconds' exposure to the heat, the muffle is narrow—some 6 to 10 in. in width, inside.

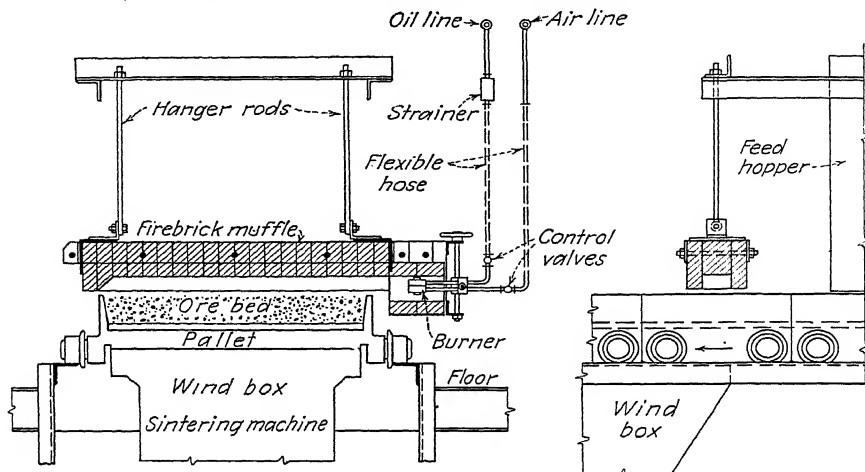


FIG. 7.—Muffle for ignition with oil.

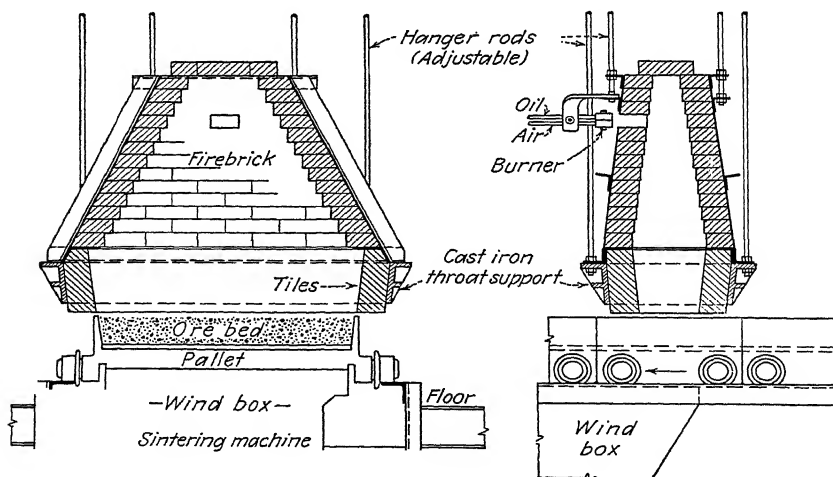


FIG. 8.—Muffle for ignition with oil.

For use with crude oil, a muffle such as shown in Figs. 7 and 8 is frequently used, the oil burner being placed at or just inside the combustion chamber. Oil and air are supplied from storage tanks and compressor through suitable piping, with regulating valves located at a point convenient for the operator.

Oil Burner.—A widely used burner is shown in Fig. 9. It consists of two small cylindrical steel blocks each having an annular groove in one flat surface, with a tapped hole for pipe connection leading to the groove at one point, and diametrically opposite a very shallow fan-shaped slot. The two are bolted together with a copper gasket between. Oil at a pressure of 20 to 60 lb. is admitted to the one half and the air for combustion, also under high pressure, to the other half. The fan-shaped jet of high-pressure air striking the jet of oil coming from the upper slot thoroughly atomizes and mixes with the latter, causing complete and almost explosive combustion. This type of burner (of which there are several models) functions independently of any combustion chamber and gives intense heat with low oil consumption. From $\frac{1}{8}$ to $\frac{3}{8}$ gal. of fuel oil is used per ton of lead or copper ore, slightly more for iron ores, which require a higher ignition temperature. Any grade of oil, from gasoline to tar, can be used in its proper burner, a distillate of 40 to 45°Bé. being the cheapest, as a rule.

For burning solid coal the same principles are followed, although the details of construction are quite different, due to the nature of the fuel. An enclosed firebox

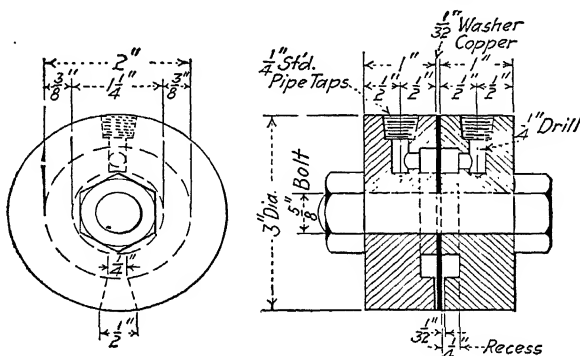


FIG. 9.—Oil burner.

is hung over the pallets at the front edge of the wind box, the hot gases from which are deflected by a bridge wall downward into a chamber corresponding to the muffle and impinge against the surface of the charge to be ignited. A dampered stack is provided for use in starting the fire before the suction fan has been started. Such a coal igniter is shown in Fig. 5.

Gas Burner.—For burning gas, two types of apparatus are used. In one a gas burner with air and gas connections, such as the Fulton blowpipe, is inserted in one end of the muffle, much as in the case of the oil burner. In the other, air and gas are caused to mix and burn in a long, narrow chamber above and separated from the muffle by a perforated plate or tile. This type is commonly used with blast-furnace gas. Whatever fuel is used, exposure of the charge to the heat is brief, as ignition is practically instantaneous, and from this point the internal combustion of the fuel in the charge maintains the reaction and supplies the heat for sintering.

Grate Bars.—The open bottom of the pallet casting is fitted with grate bars on which the charge rests. Malleable iron is the best all-round material for these bars, as it stands the alternate heating and cooling and the chemical action better than cast iron or steel, although under favorable conditions cast-iron grate bars carefully annealed have at times proved more economical than the malleable iron, in spite of

the shorter life. Bars with herringbone slots, or with straight slots from which the sinter cake pulls off readily, are used where the charge is favorable. But where easily fusible material is sintered, it tends to fill the grate-bar slots and gradually choke them. To avoid the labor of cleaning slots, various types of self-cleaning grate bar have been designed, the best known being that designed by Stewart at Trail, B.C. This consists of three stationary bars on each pallet with finger-shaped crossbars, between which are fitted two loose bars capable of slight vertical motion and with fingers which interlock with the stationary fingers. As the pallet drops over the discharge curve, the movable bars shift slightly, freeing the slots from any sinter that may have formed in them.

Automatic Proportioning and Preparation of Charges.—Placed above the drive end of the sintering machine are a superhopper and a mixer. The ore charge is brought up from the storage bins by belt conveyer or other means, to the superhopper placed in the top of the building. From the bottom of this, an automatic feeder delivers it to the mixer. This feeder is driven from the drive shaft of the sintering machine so that, as the speed of the machine is varied, the speed of the feeder automatically follows. In this way the correct amount of charge is constantly and automatically assured, irrespective of variations in the speed of the sintering machine. An adjustable gate at the hopper permits the primary adjustment of the feed. At the mixer, the charge is turned over and over, and the necessary proportion of water incorporated, until the whole is well and uniformly mixed. The proportion of water varies with the material from almost nothing to 15 to 20 per cent or even more in extreme cases. It is controlled through a valve in the water line placed on the operating floor.

The feeder and mixer may be combined in one machine, as in the round-table or revolving-plate feeder. This is similar to the Challenge feeder, and consists of a disk of steel plate, or a casting, set under the hopper. A fixed rabble arm above serves to turn the ore over as the disk revolves, while water from a spray valve moistens the material. Another type is the revolving inclined barrel, similar to the trommel. Either of these mixers works well with a material that is not sticky or otherwise difficult to mix together, such as granular ores or concentrates. But when it is necessary to mix such a thing as claylike "blue billy" with 6 per cent of fine coke, these appliances will not give a sufficiently uniform product for satisfactory sintering. For materials of this class, the old-fashioned pug mill has proved best adapted, in spite of its higher power consumption and wear. In the case of fine flotation concentrates, the cylindrical "pelletizer" hereinafter described is especially effective (see Fig. 17).

From the mixer, the charge is dropped gently through a distributing chute onto the grates. Sometimes a chute that swings from one side of the pallet to the other is used to distribute the charge evenly across the pallet back of the scraper plate, and at other plants a chute formed of a segment of a cone is used. The ore stream strikes the apex of the cone and is divided and spread in a thin sheet evenly across the pallet. This chute works well, except with sticky materials, when the swinging chute is preferred. The object is to deposit the charge on the grates in such a manner that the bed will be uniform, noncompacted, and as permeable as possible to the air that must be drawn through the bed to roast and sinter the material.

The rate of flow of the material is so regulated that a low pile of charge is maintained back of the scraper plate, which is set to maintain the bed at the desired depth. The top of the pile is only a few inches above the bottom of the plate, so as to avoid pressure. As the charge drops from the chute onto the pile, the coarser materials roll down the slope toward the approaching empty grate, and thus come to rest directly on the bare grate bars, while the finer particles are deposited on top of this as the

moving grate carries the coarse stratum underneath. As a result, the coarse supports the fine, and practically no dust is drawn into the wind boxes, even with charges as fine as flotation concentrates. A slot in the dead plate directly under the feed hopper prevents even small quantities of fines being carried over into the wind box. By maintaining the pile at a uniform height, all portions of the bed on the pallets are compressed uniformly, and therefore offer substantially the same resistance to the air blast. This ensures uniform speed of sintering, where otherwise one portion might be finished before another, on the same pallet, and the speed of the machine would have to be reduced to suit the slowest portion or else part of the charge would be discharged too soon, in either case resulting in a loss of capacity.

The mechanism for making up and handling the charge has become well standardized. Hopper-bottom storage bins, the number and size depending on the quantity and variety of materials to be sintered, are provided, each bin having an adjustable gate and a feeder of the revolving-plate or endless-belt type, which delivers a con-



FIG. 10.—Automatic proportioning of ingredients.

tinuous thin stream of ore onto a collector belt common to all bins. The metallurgist figures his charge and regulates the gate of each bin to feed the desired proportion of that material (Fig. 10). Consequently, a series of thin streams of the various materials which are to make up the desired charge is continuously deposited in successive strata on the conveyer belt so that a minimum of mixing will produce a uniform mass to be delivered to the storage hopper, commonly known as the superhopper. The feeder mechanism at the main storage bins and also the feeder at the machine hopper should be directly coordinated with the pallet speed of the sintering machine, so that the delivery of material is automatically adjusted to the requirements of the machine as its speed is changed.

From the superhopper the materials, already partially mixed, are delivered to the mixer at a rate proportional to the speed of the sintering machine. Here the necessary proportion of moisture is added and the whole thoroughly turned over to assure uniformity. The mixture now drops through the distributing chute and falls automatically in a thin uniformly porous bed onto the pallets. As the pallets are pushed forward, the top surface of the charge is momentarily exposed to the hot gases from

the igniter to start combustion, which is then continued by the air currents as the pallet travels forward over the wind box. As soon as the sintering is complete and the pallet has reached the end of the upper track, the pallet drops its cake down a chute into a bin or railroad car, and returns along the lower track to repeat the cycle. The discharge end of the sintering machine is generally placed outside the building in order to keep dust and gases away from the working place.

Each step in the process, from the making up of the charge at the storage bins to the delivery of the sintered product, is accomplished mechanically and automatically, and all the fundamental factors of successful sintering, such as uniformity of mixture, fuel content, moisture, thickness of layer, impact, porosity, intensity of ignition, air supply, both pressure and volume, and time of sintering, have been translated into a series of mechanical controls which are at the command of a single operator.

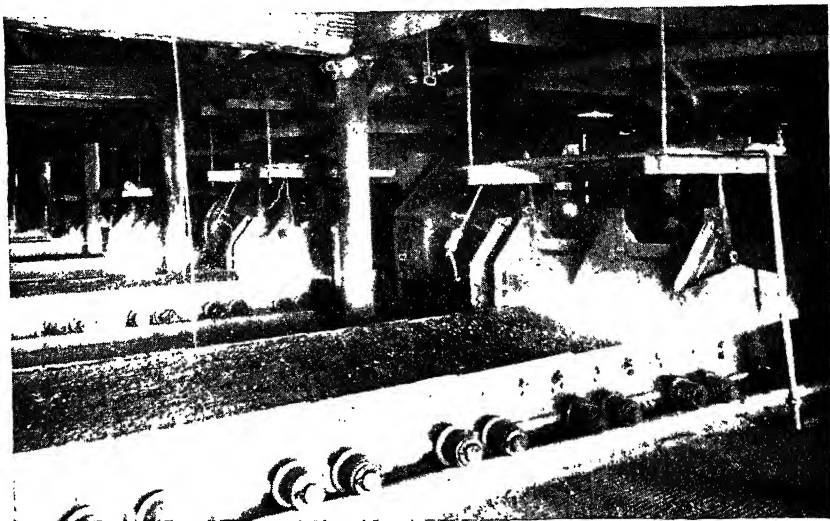


FIG. 11.—Operating floor of a Dwight-Lloyd plant.

It is the great advantage of the continuous process of sintering, compared with the intermittent, that all the conditions necessary for favorable results can be maintained with comparative ease after being once correctly established instead of having to reestablish these conditions for every separate batch. Thus the highest efficiency with the minimum of attendance may be secured.

Aside from the later types and designs of sintering apparatus, the fundamental elements of the D & L system of sintering were covered by basic U.S. patents including (1) the general "downdraft" principle which made possible successful treatment of thin layers of materials; (2) the process and rudimentary apparatus for continuous sintering; (3) the automatic proportioning, mixing, and conveying the ingredients of the charge in a continuous manner and exactly coordinated with the speed of the sintering unit; (4) the methods of ignition; and finally (5) the sinter product. These basic patents have now expired.

Sintering Practice.—The underlying principles are the same in sintering all kinds of materials, but the operating details will naturally vary somewhat. The bed of

charge on the grates is always kept shallow, the exact depth varying somewhat, but never being allowed to reach the point where the weight of the charge will compact the lower layers and destroy the characteristic cellular structure of the product. The most common depth is 4 to 5 in. with lead and copper ores, and 8 to 10 in. with iron ores. Granular material, such as jig concentrates, will permit a deeper bed than flotation concentrates. There seems to be no limit to the fineness of grain that can be sintered, provided only that the material can be laid down in a bed permeable to air blast. To this condition the proper moistening, mixing, perfectly even distribution of the charge, and freedom from compression as deposited on the pallets are of the *utmost importance* for making good sinter. At the other end of the scale, $\frac{1}{4}$ or, rarely, $\frac{1}{2}$ in., is the largest material used, except in the case of material applied as a grate dressing.

The proportion of moisture necessary varies from 3 or 4 per cent with a granular material to 20 per cent or even more with a plastic material, such as limonite. The test is to squeeze a handful of the moist charge lightly, when it should stick slightly together, or "ball." So moistened, and if not packed by excessive weight above it, a bed of such fine material as flotation concentrates will be surprisingly porous.

The rate of sintering varies from 8 to 20 min., with perhaps 12 min. as a fair average, although these are not to be taken as hard-and-fast limits. A loose, porous charge sinters more rapidly than a tight, closely packed one. And as the speed and capacity of the sintering machine are limited by the time in which the layer of material sinters, anything that will reduce this time will allow a greater pallet speed and thus increase the possible capacity of the machine. Aside from the chemical character of the charge, its rate of sintering may be said to depend on the rate of the internal combustion of the fuel in it. The greater the amount of air that can be supplied per minute, the more rapidly will the fuel burn and the faster will be the rate of sintering, always remembering that it is possible to overdo a good thing and to build up a temperature that will fuse the charge, stopping the action by blocking off the air and resulting in poor product. It is the volume of air drawn through the bed, not the vacuum used, that affects the rate of sintering, and it is the permeability of the bed, its porosity, on which air volume largely depends. A high degree of porosity—which must be uniform at all points across the pallet to permit uniform rate of sintering—is the condition aimed for. This is secured by properly moistening, mixing, and distributing the charge on the grates, and by carefully adjusting the materials going into the charge, so far as the ore supplies available to the metallurgist permit him a choice. With the continuous machine and the chain of auxiliary apparatus that has been developed to serve it, these factors are each subject to the instant, close adjustment of the operator, and, once adjusted, are maintained automatically.

Frequently, screened sinter of half-inch size or smaller ("returns") or granular ore is added to a very fine material to make it more permeable. Sometimes a thin layer of crushed limestone, sinter returns, or coarse oxide ore is fed upon the grates from an auxiliary hopper in order to prevent very fine material being drawn through the slots or to protect the grates from a very fiery charge, but this is rarely necessary.

There is a surprisingly small metal loss from dusting or fuming with a properly constituted charge. The moistened mass of fine particles forms little arches over the grate-bar slots, resisting the tendency to be drawn through by the air currents. Under normal operating conditions, not more than 1 per cent of the charge should be carried down into the wind box. With careless operating, allowing uneven distribution of the charge on the grates, improper moisture, and working with broken grate bars, the accumulations in the wind box may be considerable. The wind boxes should be cleaned out periodically, and the material sent back for retreatment.

The metal loss by fuming is very low on account of the short time that a given particle of ore is undergoing the heat-treatment. Whereas in the case of hearth roast-

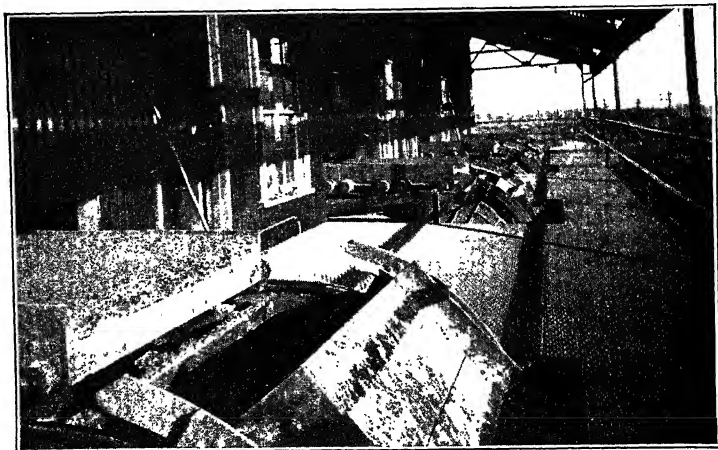


FIG. 12.—Discharge ends of row of Dwight-Lloyd sintering machines.

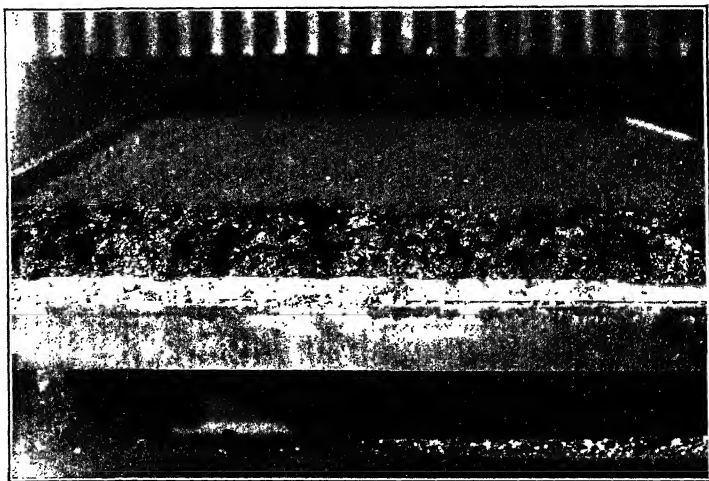


FIG. 13.—Sinter cake at discharge end of machine.

ing hours, in the case of the D&L process it is a matter of minutes.

The suction needed varies considerably, but should always be kept as low as possible, consistent with drawing the required amount of air through the bed, in order

not to compress the charge. From 5- to 10-in. water gauge is the average, except in treating iron ores, when up to 15 in. is sometimes used. But wherever possible to do so it will be found more satisfactory to increase the permeability of the bed, as by mixing in "returns," than to use a high vacuum. The amount of air required is usually 40 to 100 cu. ft. per min. per sq. ft. of hearth area (at 20°C. and 76 cm. pressure). The temperature of the waste gases is not high, usually 125 to 200°C., because, except near the far end of the wind-box section, the lower portion of the bed is not yet undergoing sintering, but is absorbing heat from the gases.

The capacity of any sintering machine is proportional to its hearth area (area of wind box). The widely used 42 × 264-in. machine, having a hearth area of 77 sq. ft.,



FIG. 14.—Dwight-Lloyd sinter.

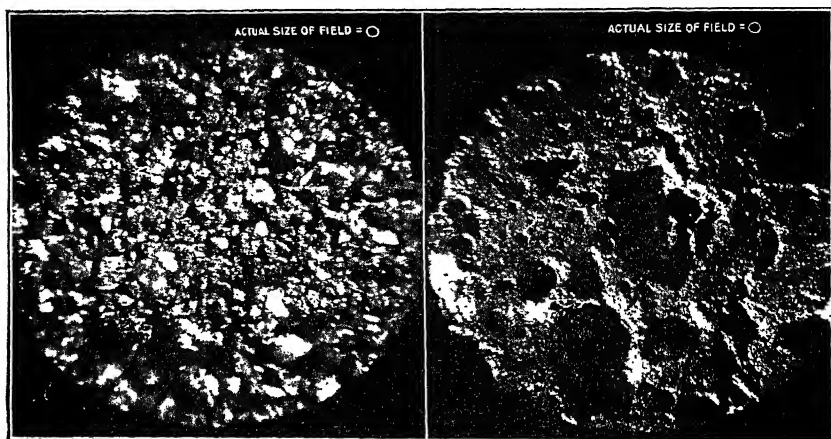
has a nominal capacity of 100 tons per 24 hr., but the tonnage actually treated is generally greater. With lead ores, single sintering, it will vary from 100 to 250 tons per 24 hr., with copper ores from 100 to 125 tons, and with zinc ores from 75 to 100 tons. Such a machine was long considered standard, especially by the lead smelters, and still presents many advantages for general work by its simplicity, efficiency and low cost of repairs.

But as the tonnages to be treated have increased, especially in the iron-ore field, the industry has gradually demanded larger capacity machines. This demand has been met by enlarging the wind-box area and (1) widening the pallets and wind box, (2) increasing the length of the wind box, or (3) increasing both width and length. In modern practice, some machines with pallets as wide as 72 in. and lengths over 100 ft. are in use.

In the writer's experience, increasing the length is preferable to increasing the width, as the weight of the pallet increases out of proportion to the increase of width, and the greatly increased shock when the pallet drops in discharging requires much heavier construction throughout. There is more grate-bar trouble and more difficulty in securing an even lateral distribution of the charge, so necessary to good work.

In handling a large daily tonnage there are advantages in dividing it over a number of smaller units compared with one large unit, with little if any difference in the labor required, and less interruption of output when one of the units requires repairs. As there is no brickwork to be heated up, operations can start at a moment's notice, so that the work can be done in one or two shifts as economically per ton as on a 24-hr. program.

But little labor is needed for operating a well-designed sintering plant. A man at the mixing bins, while making up the charges, an operator, and a helper at the machine will take care of several machines, with a day crew to unload ore into the mixing bins.



Figs. 15 and 16.—Left to right: Photomicrographs of copper flue dust before sintering; and sinter from same.

At one modern plant having four machines, three men per shift sinter 600 tons of ore per 24 hr.

The foregoing discussion of operating conditions will show that the cost of sintering varies considerably. A dollar per ton of sinter would perhaps be a fair average for a small plant, although on a larger plant costs of 50 to 60 cents are more representative.

Double Sintering.—In the blast-furnace treatment of rich lead ores, particularly where there is much zinc, it is desirable to have as little matte as possible, so that a furnace charge low in sulphur is required. To secure a thorough elimination of sulphur from these ores, the practice of double sintering was developed. The crushed sulphide charge, containing, say, 25 to 35 per cent lead and 11 to 15 per cent sulphur, is first passed over the sintering machine at a rapid rate—5 to 7 min., corresponding to 250 to 300 tons per 24 hr. on a 42 × 264 in. machine—reducing the sulphur to 6 to 7 per cent and making a partially sintered and friable cake. This is crushed in suitable crushers to $\frac{1}{2}$ to $\frac{1}{4}$ in. size, moistened, and again sintered, this time more slowly—15 to 20 min., corresponding to 100 to 125 tons per 24 hr.—and making a firm cellular cake containing only 1 to 2 per cent sulphur, which goes to the blast

furnace. The first pass is sometimes made with a 6- to 7-in. bed and a higher vacuum. Very light ignition should be used to avoid overburning or fusing the top surface. No effort to secure a sinter is made, the sole object being to remove sulphur as rapidly and cheaply as possible. On the second pass, the standard $4\frac{1}{2}$ -in. bed with lighter vacuum, but more intense ignition (on account of the low S content of the charge), is used. This charge sinters more slowly, as there is little fuel available for the reaction, but gives an excellent cake.

Formerly, at some of the plants having both H&H and D&L equipment, one stage of the double treatment was carried out in the H&H pots, but even this use of the pots has now generally succumbed to the more economical continuous machine.

Preparation of Flotation Concentrates.—The decade just passed has witnessed an enormous growth in the use of the flotation process for the beneficiation of low-grade ores of all the metals, and this has presented new problems to the metallurgists. The resulting concentrates are so fine that even in a thin layer they are almost impervi-

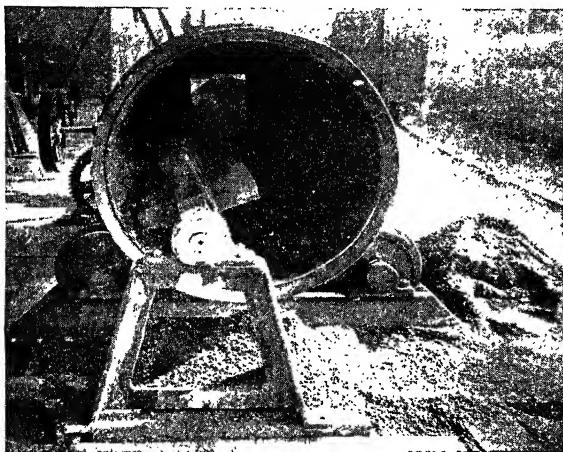


FIG. 17.—Pelletizer for flotation concentrates.

ous to air currents. The problem of sintering them is how to make a layer of them uniformly pervious. If we can do that, the rest is easy. When they contain an appreciable amount of residual oil, the difficulties of treatment are increased; the ignition is slow and the sintering action sluggish, with the capacity reduced in consequence. Probably the oil films make a tighter contact between the particles, thus impeding the air passage.

An ideal condition is created by steaming the concentrates and rubbing them through a $\frac{1}{8}$ -in. screen forming a mass of pellets. These tend to take a partial "set," so that the pellets maintain their separate shape under ordinary handling, and a mass of them when fed to the sintering machine makes an ideal charge. The individual pellets being loosely compacted are themselves relatively permeable and roast more speedily and thoroughly than ore particles of equal size. These pellets can be produced by a rolling action, but the trouble is that large dense balls form as well as small pellets and these lumps cause irregularities and poor sinter.

When received from the mills, flotation concentrates usually contain many large wet lumps which must be screened out and pulverized before they can be handled

through the feeding mechanism. The pug-mill type of mixer is well adapted to break up the lumps, though it, too, is not infallible, as balls of concentrates may form and work along to the discharge end. To meet this difficulty an improved form of pug-mill has been designed,¹ which consists of a slowly rotating drum with an inner paddle shaft, offset to one side, eccentric to and below the axis of the drum. The rapid action of the paddles thoroughly mixes and "fluffs" the charge, while any lumps that may be present are carried into contact with the paddles by the rotation of the drum and broken up. Figure 17 shows a pelletizer mill of this type with some of the product.

Another method of sintering flotation concentrates which is widely used is to mix with the raw charge a proportion of finished sinter crushed to minus $\frac{1}{2}$ or $\frac{1}{4}$ in. size. These "returns" serve two purposes in the case of sulphide ores. Being low in sulphur they dilute the sulphur content of the mixture so that a single sintering operation produces a low sulphur in the sinter cake. As the returns are comparatively coarse, they loosen up the mixture and make the charge more permeable to air, while at the same time each piece of returned sinter becomes an inert nucleus coated with a thin layer of the sticky fine sulphide particles which are thus presented in an ideal way for rapid and efficient blast roasting. The result is that the operation is rapid, and a much larger tonnage can be treated with a given equipment than would otherwise be possible. A small proportion of granulated slag has been successfully used in place of returns. In sintering flotation concentrates from oxidized ores, returns may also be used to open up the charge with considerable advantage.

More care in the preparation of the charge for sintering and a very thorough mixing of the ingredients is required in this procedure than in ordinary sintering or double sintering, for if large lumps of concentrates are permitted to get into the feed, or if the coarse material is not evenly distributed through the fines, the results will not be good, as the sulphur will be high.

Applications of Blast Roasting.—As previously stated, many applications of the downdraft blast roasting principle has been found which were not originally contemplated, and some of these do not, strictly speaking, include actual roasting. As the apparatus used is the same, and the variations in the technique are often so slight that a casual observer would have difficulty in telling what class of material was being treated on the sintering machine, it is but natural that the designation "blast roasting," as originally applied, should be used in a broader sense to cover this general method of heat-treatment by internal combustion (page 301).

Where the material contains naturally no fuel constituents, such as sulphur in sulphide ores, the deficiency can be made up by an admixture of carbon in the form of coke fines or small anthracite coal, the former being the more suitable. In sintering fine iron ores, the proportion of mixed fuel varies from 5 per cent or less with some magnetites to 8 per cent with materials where the iron is present as Fe_2O_3 , as in pyrites cinder. Bituminous coal can be used with precautions, but is not desirable for this purpose on account of the accumulation of tarry products in the flues and fan and the possible danger of explosions of the unused and unburned volatile hydrocarbons. Similarly, peat has been used with reasonable success.

The uses that have been found for the D & L process including the following:

1. Roasting and sintering of ores and by-products preparatory to lead blast-furnace smelting. This has now become standard practice the world over. In many plants the entire blast-furnace charge, including fluxes, is made up by the D&L automatic system and put over the sintering machine on its way to the furnaces. This has resulted in great economies in labor, metal losses, coke consumption, and furnace speed.

¹ Sintering Machinery Corp., Netcong, N. J.

2. Roasting and sintering of copper sulphide ores preparatory to blast-furnace smelting. By careful manipulation it is possible to minimize the expulsion of sulphur, where conservation of the available sulphur is important, as is often the case. On account of the great improvements made in reverberatory smelting in the past decade, sintering has not found the same field of usefulness in copper smelting that it has in lead smelting.

3. Sintering of fine iron ores, concentrates and sulphur-bearing iron ores heretofore unusable, and the reclamation of iron blast-furnace flue dust and pyrites cinder, which were once considered waste products. This has found wide application in American iron practice. Furnaces have been run on 100 per cent D & L sinter with manifest operating advantages and remarkable saving in coke.

4. Sintering of ores and concentrates of blende, after a preliminary roast down to 8 or 10 per cent sulphur, which can be accomplished in a suitable furnace without extraneous fuel, and finishing on the sintering machines down to under 1 per cent sulphur.¹ It has been found that the product yields a larger direct zinc output from the retorts with a distinct saving in fuel. This practice has been adopted by a number of zinc smelteries in the United States.

5. Agglomeration of small scrap metal of many kinds, jeweler's sweepings, etc., in metal-reclamation plants. This use is quite extensive.

6. Other special uses have been found in chemical technology, where a heat-treatment with or without sintering is desired, as, for example, calcination of lime and other earthy materials, cement clinkering, and the treatment of phosphate rock.

7. Sintering dolomite for lining openhearth steel furnaces.

8. Preparing sinter for use in openhearth steel furnaces in place of iron and steel scrap.

TABLE 1.—IGNITION TEMPERATURES OF SULPHIDES

Material	Size of grain, millimeters		
	1	1-2	Over 2
Pyrite.....	325°C.	405°C.	472°C.
Pyrrhotite.....	430	525	590
FeS.....	...	535	
Ni 73.3 S 26.7	700	802	886
Co 66.37 S 33.63	574	684	859
Co 70.20 S 29.80	514	751	1,019
Stibnite.....	290	...	340
Molybdenite.....	240	...	508
Cinnabar.....	338	...	420
Chalcocite.....	430	...	679
Bi 83.3 S 16.7	500	...	626
Mn 61.01 S 33.98 Fe 2.02.....	355	...	700
Argentite.....	605	...	875
Blende.....	647	...	810
Galena (in oxygen).....	554	...	847
Millerite.....	573	...	616

¹ U.S. patent 1401733, by Gilbert Rigg.

CHAPTER XI

REFRACTORIES

BY FRED A. HARVEY¹

General Aspects.—The term “refractories” embraces all materials used in the arts for the construction of heat-resisting containers, using the word in its broadest sense, whether it be to afford space for the evolution of gases in combustion processes or the holding of molten charges or of solids undergoing heat-treatment. The two principal functions involved in the use of refractory materials are those of thermal insulation and conduction. In the outside walls of a furnace, refractories serve the purpose of confining the heat and preventing an excessive loss to the atmosphere; in a muffle or retort, they serve to conduct the heat through the walls to the charge.

Although, in general, the heat-resisting quality of refractories is of paramount importance, this is by no means the only requirement and may at times be only a secondary consideration. Refractories may be expected to be relatively unaffected by high temperatures under stress whether negligible or heavy; to resist mechanical abrasion at various temperatures; to resist the intrusion of molten metals, slags, or metallic vapors, as well as the action of superheated steam and hydrocarbons, sulphurous oxide, chlorine, or other gases; and to withstand sudden temperature changes. Under one set of conditions, high thermal conductivity may be required and under another high insulation value, while in still other cases high electrical resistance at moderately high temperatures may be demanded. It is obvious that no single refractory will completely fulfill all these functions, and hence the proper selection of a suitable material is often a complicated task.

Since in many reactions the yield or speed increases rapidly with increase in temperature above a given point, it is clear that a gain in working temperature of only a few degrees, made possible by the use of a more refractory material, may result in so much higher returns that the additional cost of the installation is unimportant.

General Properties of Refractories.—It is generally realized by experienced operators that the virtues and faults of refractories for a given purpose are usually determined by one or two outstanding properties possessed by a material, other considerations being of secondary importance. Not infrequently a refractory that is rated low from the standpoint of heat resistance may be the most economical material for some special service. For this reason it is desirable to consider the principal chemical and physical properties called into play in service and to discuss them for each type of refractory. Moreover, the methods employed for estimating these special properties should be considered, since familiarity with them will make it possible to differentiate the various materials and to recognize both their advantages and their limitations.

The principal factors involved in the use of refractories may be summarized as follows:

1. Chemical and mineralogical composition.
2. Refractoriness.
3. Permanent expansion or contraction upon reheating.
4. Porosity and specific gravity.

¹ Late Director of Research, Harbison-Walker Refractories Co., Pittsburgh, Pa.; deceased April 27, 1945.

5. Resistance to compression at furnace temperatures.
6. Thermal properties: heat capacity, thermal conductivity, and expansion.
7. Resistance to rapid temperature changes. Thermal spalling.
8. Resistance to slagging action.
9. Mechanical strength and resistance to abrasion.

These factors are not necessarily arranged in their order of importance; in fact they cannot be so arranged, since one property, such as resistance to spalling, may be most important in one furnace, while another property, such as low porosity, may be far more important in another type of furnace.

Chemical and Mineralogical Composition.—The classification of refractories for metallurgical purposes has been based frequently on the composition of the materials as expressed by their acidity, *i.e.*, the ratio of silica to the basic constituents. It has been assumed that siliceous slags would not attack refractories high in silica, and that basic slags would not combine with basic refractories. In a general way, this assumption is undoubtedly true, but it must be realized that the resistance of a refractory to attack by slag is influenced to a marked degree by its density. Porous refractories, no matter how well adapted they may be from the chemical standpoint to resist attack by certain slags, are poorly suited for such requirements, since they absorb liquid slag by capillary action and hence may be destroyed in a short time. Again, in the calcination of solids, an excellent showing may be made by refractories whose chemical characteristics are exactly opposite to those of the charge. In the burning of lime for example, silica-brick kiln linings give excellent results. This is due to the fact that the temperature of calcination is below the formation of any lime-silica eutectic. If the temperature rises above the eutectic point, about 2650°F, a destructive reaction occurs.

As the temperature is raised, the reactions between solids occur at progressively higher rates and diffusion phenomena are more rapid. In the openhearth furnace, therefore, it is usual to separate the acid silica brick from the basic magnesite brick by means of a neutral course of chrome brick, a practice also followed in many copper reverberatory smelting furnaces.

It is evident that chemical composition is of fundamental importance in determining the character and the quality of a refractory. A low content of basic impurities in siliceous materials and of acid impurities in basic refractories is obviously desirable, and for this purpose chemical analysis is useful.

Committee C-8 of the A.S.T.M. has developed accurate methods for the chemical analysis of the common refractory materials.¹

A rough classification of refractories is *Acid*: silica sand, quartz, ganister, quartzite, fire clays, flint clays, kaolin, schists; *Neutral*: bauxite, diaspore, fused alumina, chromite, zirconium silicate, silicon carbide, kyanite, spinels, corundum, magnesium silicate, graphite; *Basic*: magnesite, lime, dolomite, magnesium ferrite, and zirconium oxide.

The mineralogical structure of the heat-resisting materials is of great importance in the study of this subject. The principal tools for studying mineral compositions are the petrographic microscope and X-ray equipment. With these, it is possible to acquire a thorough knowledge of the conversion of quartz to cristobalite and tridymite in silica brick, the formation of mullite in clay firebrick, the crystallization of spinels, the recrystallization of periclase in magnesite brick, and the formation of accessory minerals.

¹ These methods appear in "Manual of A.S.T.M. Standards on Refractory Materials," pp. 59 to 85, 1941 edition. They may be obtained from the society and are known as A.S.T.M. Designation C 18. All the A.S.T.M. Standards on Refractories have been published in this manual. It includes also several valuable surveys of conditions of refractory service in industrial furnaces.

Refractoriness.—It is evident that a refractory retains its usefulness only as long as it continues to be sufficiently rigid to support its own weight and such loads as may be imposed in service.

Unfortunately, the term "refractoriness" has no specific meaning. For many years it was loosely considered to represent the so-called melting point, or fusion point, of the material.¹

Such a definition introduces a serious difficulty, as most of the common refractory materials possess no melting point in the proper physical conception of this term. Pure alumina, Al_2O_3 , and pure forsterite, $2\text{MgO} \cdot \text{SiO}_2$, are exceptions. Alumina melts at 3704°F . and forsterite at 3470°F .

It is well known that silicates, such as clays, offer no definite transition point from the solid to the liquid phase. It is necessary, therefore, to depend on deformation data, such as the bending of specimens of standard size and shape in the manner of Orton cones, or a load test. Even when a standard condition of deformation has been established, it is evident from theoretical considerations that the rate of heating will have a decided influence on the deformation temperature. Rapid firing results in a higher temperature of deformation than a slow rate of heating. This condition is made still more complicated by the heterogeneous nature of the materials to be tested, which consist of particles of different substances varying greatly in size. Thus, the determination of refractoriness by deformation is not so simple as it might appear. It undoubtedly is true that many more failures occur because of softening and shrinkage than are due to true fusion.

The same indefiniteness is found in those refractories whose failure is due to decomposition rather than plastic flow. There is usually no definite point at which decomposition begins, although it becomes much more rapid with increasing temperature.

The A.S.T.M., through its Committee C-8 on Refractories, has coined the term "pyrometric cone equivalent" (P.C.E.) and established standard Method C 24 for the determination of this property. Pyrometric cone equivalent is defined by A.S.T.M. as follows: In the case of refractories, the number of that standard pyrometric cone whose tip would touch the supporting plaque simultaneously with a cone of the material being investigated when tested in accordance with the Standard Method of Test for Pyrometric Cone Equivalent of Refractory Materials (A.S.T.M. Designation C 24) of the American Society for Testing Materials.

Pyrometric Cone Equivalent (P.C.E.) of Refractory Materials (A.S.T.M. C 24).

1. This method of test covers a procedure for determining the pyrometric cone equivalent of fire clay, fire-clay brick, silica cement, and ground fire clay by comparison of test cones with standard pyrometric cones under the conditions herein prescribed.

Preparation of Sample

2. (a) *Lump Clay or Brick.*—The entire sample of lump clay or brick, in case the amount is small, shall be reduced in size by means of rolls or a jaw crusher to a maximum size of $\frac{1}{4}$ in. In case the amount is large, a representative sample obtained by approved methods shall be so treated. The sample shall then be mixed thoroughly and the amount reduced to about 250 g. (0.5 lb.) by quartering. Any magnetic material introduced by crushing or grinding shall be removed by means of a magnet. This portion shall then be ground in an agate, porcelain, or hard steel mortar and the amount reduced again by quartering. The final size of the sample shall be 50 g. and the fineness such that it will pass a No. 70 (210-micron) A.S.T.M. sieve² (equivalent

¹ HARVEY, Pyrometric Cone Equivalent vs. Fusion Point, *Proc. A.S.T.M.*, Vol. 34, Part I, pp. 376–377, 1934.

² This and other A.S.T.M. methods are reproduced with permission of the American Society for Testing Materials.

to mesh No. 65 of the Tyler standard series). In order to avoid excessive reduction of fines, they shall be removed frequently during the process of reduction by throwing the sample on the sieve and continuing the grinding of the coarser particles until all the sample passes through the sieve.

(b) *Fire-clay and Silica Cements*.—In the case of ground fire clay and silica cements, the sample obtained by approved methods shall be tested as received without grinding or other treatment.

NOTE:—Neither fire clay nor silica cements include the materials sold under the name of high-temperature bonding mortars.

Preparation of Test Cones¹

3. (a) The dried sample shall be thoroughly mixed and after the addition of sufficient dextrin, glue, gum tragacanth, or other alkali-free organic binder and water,

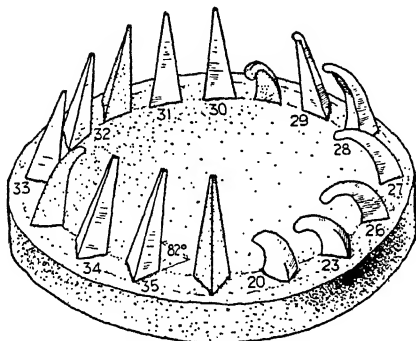


Fig. 1.—Method of mounting test cones and appearance after testing.

shall be formed in a metal mold into test cones in the shape of tetrahedrons 8 mm. ($\frac{5}{16}$ in.) on the sides at the base and 25 mm. (1 in.) in height.

(b) When dry the test cones may be subjected, when necessary, to a preliminary burn at a temperature not exceeding 1300°C. for the purpose of sintering them to a firm condition to permit handling.

Mounting

4. The test cones shall be mounted on plaques of refractory material of such a composition as will not affect the fusibility of the cones.² They shall be mounted with the base embedded approximately 3 mm. (0.12 in.) in the plaque and the face of one side inclined at an angle of 82 deg. with the horizontal. The arrangement of the test cones with respect to the standard pyrometric cones shall be substantially as shown in Fig. 1, *i.e.*, alternating the test cones with the standard pyrometric cones in-

¹ Detailed requirements for this sieve are given in the Standard Specifications for Sieves for Testing Purposes (A.S.T.M. Designation F 11) of the American Society for Testing Materials; see 1939 Book of A.S.T.M. Standards, Parts II and III.

² A mixture of equal parts of a highly refractory clay such as a good grade of china clay and fused alumina which will pass a No. 100 (149 micron) A.S.T.M. sieve (equivalent to mesh No. 100 of the Tyler standard series) has been found satisfactory.

sofar as is practical. The plaque may be any convenient size and shape and may be biscuited before using, if desired.

NOTE:—The number of cones and their mounting so as to face inward as shown in Fig. 1 is typical for gas-fired furnaces of relatively large dimensions and gases moving at high velocity. The practical bore of the muffle tubes in most electric furnaces does not permit cone pats of this size. The static atmosphere prevailing permits the cones being mounted to face outward, if so desired.

Heating

5. (a) The heating shall be carried out in a suitable furnace at a rate to conform to the following requirements:

To reach cone 20.....	45 min.
Time interval to:	
Cone 23.....	20 min.
Cone 26.....	6 min.
Cone 27.....	4 min.
Cone 28.....	4 min.
Cone 29.....	10 min.
Cone 30.....	4 min.
Cone 31.....	12 min.
Cone 32.....	8 min.
Cone 32½.....	9 min.
Cone 33.....	9 min.
Cone 34.....	6 min.
Cone 35.....	10 min.
Cone 36.....	10 min.
Cone 37.....	4 min.
Cone 38.....	6 min.

NOTE:—At 1000°C. the old cone pat may be removed and a new one put in without cooling the furnace to below red heat.

(b) That type of furnace in which a neutral or oxidizing atmosphere may be maintained is to be preferred. Excessive reducing conditions should be avoided. Care should be taken that the flame does not strike directly against the cone or cone plaque. The furnace should be tested at intervals for the determination of the uniformity of the distribution of the heat.

Pyrometric Cone Equivalent

6. (a) The softening of the cone will be indicated by the top bending over and the tip touching the plaque. The bloating, squatting, or unequal fusion of small constituent particles should always be reported. The pyrometric cone equivalent shall be reported in terms of standard pyrometric cones and shall be that cone which most nearly corresponds in time of softening with the test cone. If the test cone softens later than one standard pyrometric cone but earlier than the next standard pyrometric cone and approximately midway between, the P.C.E. shall be reported thus: cone 31-32. If the test cone starts bending at an early cone but is not down until a later cone, the fact should be reported.

NOTE:—The temperatures corresponding to the end points of the standard pyrometric cones are frequently of interest. These are shown below.

The temperatures corresponding to the end points of those standard pyrometric cones that are used in connection with refractories are as follows:

Cone	End point		Cone	End point	
	°C.	°F.		°C.	°F.
15	1435	2615	30	1650	3002
16	1465	2669	31	1680	3056
17	1475	2687	32	1700	3092
18	1490	2714	32½	1722 (approx.)	3131
19	1520	2768	33	1745	3173
20	1530	2786	34	1760	3200
23	1580	2876	35	1785	3245
26	1595	2903	36	1810	3290
27	1605	2921	37	1820	3308
28	1615	2939	38	1835	3335
29	1640	2984			

These temperatures (except cone 32½) which were determined by Fairchild and Peters¹ for a heating rate of 150°C. per hr. for cones 15 to 20, and of 100°C. per hr. for cones 23 to 38, other conditions being the same as specified, apply satisfactorily for all the conditions of this test method, but do not apply to conditions of the commercial burning and use of refractory materials.

In spite of the care used in establishing the procedure of method C 24, it is found that different laboratories fail to check one another closer than within about one cone. Various types of furnaces are used in making P.C.E. tests. They are heated by gas, either natural or manufactured, by acetylene, or by an electric current and carbon resistance heating elements. It is probable that the gas-fired furnace manufactured by the Denver Fireclay Co. is more widely used than any other type.

Permanent Expansion and Contraction.—Practically all refractories, when heated for long periods under the conditions of their use, are subject to changes in volume. These volume changes are permanent in distinction to reversible changes brought about by thermal expansion and contraction. Clay, magnesite, and alumina refractories tend to contract or shrink, while silica brick and a few other siliceous materials expand. It is evident that excessive volume changes are exceedingly undesirable, inasmuch as they tend to bring about deterioration of the furnace structure, through the opening up of joints and the formation of cracks, thus causing leakage or possible slag penetration. For most purposes, materials showing the least contraction or expansion in use are most desirable. Excessive shrinkage may be due to insufficient firing during manufacture or to lack of refractoriness.

A knowledge of the volume and porosity changes of a refractory raw material during firing is indispensable to the manufacturer. A method frequently used for the determination is known as the method of draw trials. By this method, a series of specimens, 1½ × 1½ in. cylinders, 6 × 1 × 5⁄8 in. bars, or small cubes, are prepared. These specimens are then fired and drawn from the furnace as definitely scheduled temperatures are reached. The specimens should be numbered with ceramic ink to ensure their identification.

The temperature range must be adjusted to the kind of refractory being tested. If a fire-clay material is being used, the range between 2100 and 2700°F. may be

¹ FAIRCHILD and PETERS, Characteristics of Pyrometric Cones, *Jour., Amer. Ceramic Soc.*, vol. 9, No. 11, p. 700, November, 1926.

chosen. The temperature is raised to 1800°F. in 2 hr., and after 1800°F. has been reached at the rate of 50°F. per hr., and this schedule must be adhered to closely in order that it may be possible to check the results in other firings. If the clay contains much carbonaceous material a hold of 4 or 5 hr. at 1500 to 1700°F. is necessary to ensure oxidation. The specimens must be so placed that they can be reached from without the furnace and withdrawn by means of suitable tongs. It is also well to place a series of pyrometric cones as close to the specimens as possible, to serve as a check on the heat-treatment. When the pyrometer indicates the temperature of 2100°F., the first three specimens should be withdrawn and at once placed in an auxiliary muffle furnace maintained at a temperature of about 1100 to 1300°F., or placed in a tray and covered with insulating powder. If the temperature interval

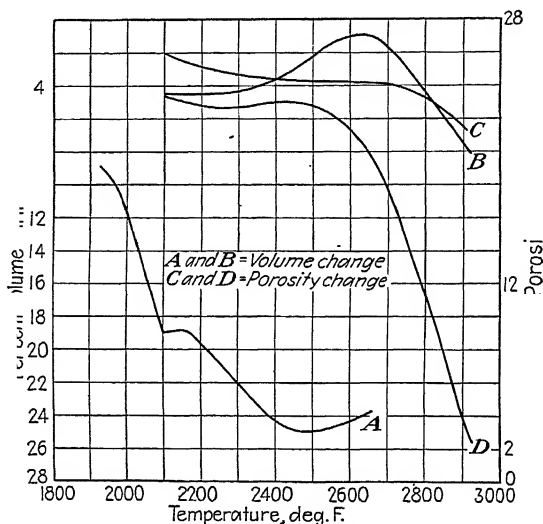


Fig. 2.—Volume-porosity changes on firing.

decided upon is 50°F., the next specimens must be removed at 2150°F., and so on, until the maximum temperature has been reached. The temperature intervals may be taken as large or as small as is convenient. Where but few points are desired, the intervals may be made as large as 100°. When all the specimens have been removed and placed in the auxiliary muffle, the latter is allowed to cool at its natural rate.

It is customary to compute the percentage of shrinkage of each specimen based upon its volume in the dried state according to the obvious relation $100(V_1 - V_2)/V_1$, where V_1 is the volume of specimen in the dried state and V_2 its volume in the fired state. The plotting of these results in graphic form offers a convenient means of comparison, from which an estimate of the behavior of the material can be readily made. Such a graph is shown in the curve A of Fig. 2 which indicates that the refractory continues to contract until the temperature 2450°F. has been reached. The material, therefore, must be burned to this point if it is desired that it be as constant in volume as possible. It will be noted that beyond this temperature the specimen expands. This phenomenon is very characteristic of clays that "overfire," i.e., on

account of excessive softening caused by heat and the evolution of gases, they develop a vesicular or spongy structure accompanied by a swelling of the mass. Clay in this state tends to flow and deform. It possesses low mechanical strength and offers but little resistance to corroding influences such as slags. The diagram thus gives important data concerning a refractory, *viz.*, the rate of shrinkage with temperature, the temperature at which maximum condensation has taken place, and the point at which "overfiring" begins.

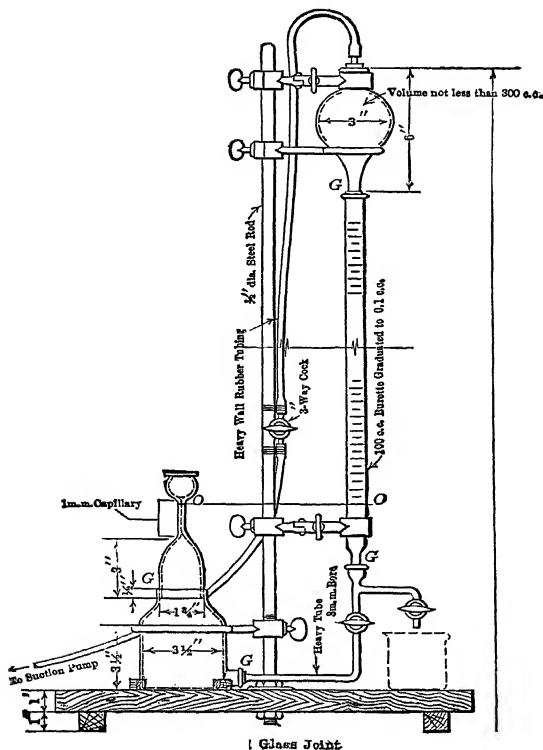


FIG. 3.—Seger voluminometer.

Many clays exhibit the property known as "secondary expansion," and this expansion must not be confused with that due to overfiring, since it does not indicate a lack of refractoriness. Curve *B* of Fig. 2 is typical of Pennsylvania flint clays that show secondary expansion. The clay had a P.C.E. of cone 33-34.

The amount of shrinkage (or expansion) will depend on the grain size of the material from which the specimens are formed, and in order to have comparable test data, the material is frequently ground so that it will all pass a 20-mesh sieve.

If the volume changes are determined by weighing dry, suspended in water or kerosene, and saturated, the same data will make it possible to calculate porosity for the various specimens. This information is frequently useful.

Instead of drawing the specimens from the hot furnace, it is sometimes desirable to make a separate furnace run for each final temperature, hold this temperature for 5 hr., and allow the specimens to cool in the furnace. This takes much longer than a draw trial method, but gives results more directly applicable to manufacturing conditions.

It is evident that such studies also show volume alterations due to molecular changes, such as the transformation of quartz to cristobalite and tridymite and the decomposition of clay into mullite and silica.

A.S.T.M. method C 113¹ covers the determination of permanent linear change after reheating of refractory brick. This method is used to determine whether the brick has been burned to a sufficiently high temperature. It may also be used for testing clays.

Equipment for Volume Studies.—The volume of the specimens may be determined by several methods, whichever may be the more convenient. The direct determination of volume is made conveniently by means of a voluminometer such as the Seger type (Fig. 3).

If the specimens are regular in contour, such as the cylinder mentioned, the volume may be determined by direct measurement and calculation.

Indirectly, volume may be determined by several weighing methods. One method consists in saturating the specimen by immersion in a suitable liquid, kerosene or water, and weighing it in this state. The same test piece is then weighed while suspended in the liquid. From these weights the exterior volume of the specimen, in cubic centimeters, is computed from the relation $v = (w - s)/d$, where v is the exterior volume, w the saturated weight, s the suspended weight in grams, and d the density of the liquid. A rapid method, requiring no previous saturation of the test piece, is to weigh in air and then find the weight necessary to submerge it in mercury.

Porosity and Density (A.S.T.M. Method C 20).

1. This method of test covers procedures for determining the following properties of burned refractory products:

Apparent porosity.

Water absorption.

Apparent specific gravity.

Bulk density.

NOTE.—This method is not applicable to materials attacked by water.

Preparation of Sample

2. (a) The sample shall consist of at least five standard 9-in. brick or similar units, from each of which a single test specimen, having a volume of approximately 25 or 26 cu. in., shall be cut or broken. When testing standard 9-in. straight brick, the specimen shall be a quarter-brick obtained by first halving the brick along a plane parallel to the $9 \times 2\frac{1}{2}$ in. face and then again halving along a plane parallel to the $4\frac{1}{2} \times 2\frac{1}{2}$ in. face. Four of the surfaces of the resultant quarter-brick specimen include part of the original molded faces. These surfaces are half of one end, half of one side, and one quarter of each of the two $9 \times 4\frac{1}{2}$ in. faces. When testing large shapes, the sample shall consist of several specimens cut or broken from each shape from both the center and outer portions.

Each specimen shall be freed of all loosely adhering particles.

Visibly defective specimens shall not be used.

¹ Manual of A.S.T.M. Standards on Refractory Materials.

Dry Weight D

3. (a) The test specimens shall be dried to constant weight by heating to 105 to 110°C. and the dry weight D in grams determined to the nearest 0.1 g.

The drying procedure may be omitted only when the test specimens are known to be dry, as may be the case with samples taken directly from kilns.

The drying of the specimens to constant weight and the determination of their dry weights may be done either before or after the boiling operation (Sec. 4). Usually the dry weight is determined before boiling; if, however, the specimens are friable or evidence indicates that particles have broken loose during the boiling operation, the specimens shall be dried and weighed after the suspended weight S and the saturated weight W have been determined as described in Secs. 5 and 6. This second dry weight shall be used in all appropriate calculations.

Boiling

4. (a) The test specimens shall be placed in water and boiled for 2 hr. During the boiling period, they shall be kept entirely covered with water and shall not be in contact with the heated bottom of the container.

After the boiling period, the test specimens shall be cooled to room temperature while still completely covered with water. The cooling may be accelerated by running cold water into the container.

Suspended Weight S

5. (a) The weight S of each test specimen after boiling and while suspended in water shall be determined in grams to the nearest 0.1 g.

This weighing is usually accomplished by suspending the specimen in a loop or halter of 22-gauge copper wire hung from one arm of the balance. The balance shall be previously counterbalanced with the wire in place and immersed in water to the same depth as is used when the refractory specimens are in place.

Saturated Weight W

6. After determining the suspended weight, each specimen shall be blotted lightly with a moistened smooth linen or cotton cloth to remove all drops of water from the surface and the saturated weight W determined in grams by weighing in air to the nearest 0.1 g. The blotting operation shall be performed by rolling the specimen lightly on the wet cloth, which has previously been saturated with water and then pressed only enough to remove such water as will drip from the cloth. Excessive blotting will introduce error by withdrawing water from the pores of the specimen.

Exterior Volume V

7. The volume V in cubic centimeters of the test specimens may be obtained by subtracting the suspended weight from the saturated weight, both in grams as follows:

$$V = W - S$$

NOTE.—This assumes that 1 cc. of water weighs 1 g. This is true within about 3 parts in 1000 for water at room temperature.

Volumes of Open Pores and Impervious Portions

8. The volume of open pores and the volume of the impervious portions of the specimen may be calculated as follows:

$$\begin{aligned}\text{Volume of open pores in cc.} &= W - D \\ \text{Volume of impervious portion in cc.} &= D - S\end{aligned}$$

Apparent Porosity P

9. The apparent porosity expresses as a percentage the relationship of the volume of the open pores of the specimen to its exterior volume and shall be calculated as follows:

$$P = \frac{W - D}{V} \times 100$$

Water Absorption A

10. The water absorption A expresses as a percentage the relationship of the weight of water absorbed to the weight of the dry specimen and shall be calculated as follows:

$$A = \frac{W - D}{D} \times 100$$

Apparent Specific Gravity T

11. The apparent specific gravity T of that portion of the test specimen which is impervious to boiling water shall be calculated as follows:

$$T = \frac{D}{D - S}$$

Bulk Density B

12. (a) The bulk density B in grams per cubic centimeter of a specimen is the quotient of its dry weight divided by the exterior volume, including pores, and shall be calculated as follows:

$$B = \frac{D}{V}$$

(b) This method of determining bulk density is useful for checking bulk density values obtained by the direct measurement method.¹ While it is more accurate than the direct measurement method, and generally gives higher values (by about 0.02 to 0.04), the direct measurement method is better suited for plant and field testing, since it is a less involved technique. The present method is preferable for specimens which are branded deeply or irregular in contour.

Report

13. (a) For each property, the average of the values obtained with at least five specimens, and preferably also the individual values, shall be reported.²

(b) Apparent porosity and water absorption results shall be reported to one decimal place, and apparent specific gravity and bulk density results to two decimal places.

NOTE.—When values are reported for water absorption but not for porosity, it is suggested that the report shall also give the results for bulk density. This makes it possible to calculate the corresponding apparent porosity values as follows:

$$P = A \times B$$

End of Method C 20.

When refractories are heated, they show reversible thermal expansion, *i.e.*, when heated they expand and when cooled they contract, just as do most materials. However, with increase in temperature above some definite point, the apparent volume of

¹ Methods of Test for Size and Bulk Density of Refractory Brick (A.S.T.M. Designation C 134) 1941 Supplement to Book of A.S.T.M. Standards, Part II.

² When there are pronounced differences among the individual values, another sample of five specimens shall be tested. These shall consist of the quarter-brick diagonally opposite from the original specimens. The average of all 10 determinations shall be reported.

most refractories decreases steadily, due to colloidal changes or to the influence of fluxes which tend to bring about softening of the mass. In general, most refractory oxide bodies undergo contraction upon being heated to a sufficiently high temperature. Contraction must in a large measure be described as a surface-tension phenomenon. This force is active in reducing the superficial area of the mass and theoretically tends to reduce the exterior volume to the shape of a sphere. At a lower temperature the rigidity of the system is too great to allow appreciable contraction, but at higher temperatures the effect becomes more and more manifest, as may be readily demonstrated by heating alumina, magnesium oxide, thorium oxide, zirconium oxide, etc., to temperatures somewhat below the temperature of complete fusion. Contraction increases rapidly in the presence of fluxes, so that, for instance, the addition of a small amount of boric acid will cause alumina to reach constant volume at 3180°F., about 520°F. below its fusion point. The presence of basic oxides in clay and siliceous materials and of acid compounds in the basic refractories tends to form eutectic combinations which depress the softening temperature of the whole and at the same time bring about a marked decrease in the viscosity of the system.

Even though the refractories are as free from fluxing materials as is commercially possible when placed in service, they frequently pick up harmful fluxes from the furnace slags or dust-laden gases. This condition may cause a marked shrinkage and vitrification of the hot face at furnace working temperatures. Vitrification makes the refractory more susceptible to spalling, and if the furnace temperature fluctuates, may cause cracks to open up, allowing the penetration of more slag. By properly choosing the refractories, and by careful bricklaying, slag penetration may be minimized.

Porosity and Specific Gravity.—Coincident with the shrinkage in volume as the temperature increases, there occurs contraction of the pore space. The reduction in porosity may, therefore, be taken as one criterion of the gradual softening of the mass, and as it is determined quite readily, it is sometimes used to establish the refractory character of a material. This method is not so sensitive with reference to small temperature effects as the measurement of volume, but is sufficiently accurate for most purposes. The determination of porosity for materials not attacked by water is covered by A.S.T.M. Method C 20. When necessary, kerosene may be used for the liquid and the equation corrected by using the specific gravity of kerosene as on page 332.

As in the case of changes in volume, the apparent porosities of a material after firing to different temperatures may be plotted against the temperature. The curves thus obtained are equally instructive and valuable for the comparison of the pyrotechnical properties of refractories. Curves *C* and *D* of Fig. 2 give the results obtained with fire-clay materials. Here again the rate of porosity decrease and the temperature at which the structure has become dense—shown by the approach to zero porosity—offer data of practical importance.

Since the determination of apparent porosity depends to a large extent upon the absorption of water by the material, it will suffice in some cases to make use of water absorption in establishing the relation between temperature and its effect upon the consolidation or vitrification of the body. The determination of water absorption is covered in Method C 20. The value merely expresses the ratio of the weight of water absorbed to the dry weight of the specimen. The apparent porosity, which is the ratio of the volume of open pores to the exterior volume, is to be preferred in all cases where close differentiation is desirable or where the comparison of bodies possessing different specific gravities is involved.

Occasionally, it may be desirable to determine the total porosity of refractories, especially when they are dense and the absorption of water is incomplete or when

closed pores exist into which the liquid cannot penetrate. For the calculation of this value, it is necessary to determine both the bulk and the true specific gravity. The former is determined by obtaining the weight of the dried piece, in air, and its exterior volume, as described in a preceding paragraph. The bulk density, obviously, is equivalent to the relation w/v , where w is the weight of the specimen and v its volume. The true specific gravity is found by grinding the material, passing it through the 100-mesh sieve, and determining the values sought by means of a pycnometer in the usual manner. The true porosity is then computed from the relation $p = 100(1 - d_1)/d_2$, where p is the per cent pore space in terms of the exterior volume, d_1 true specific gravity, and d_2 bulk specific gravity.

The determination of true specific gravity is fully covered by A.S.T.M. Method C 135.

It is interesting to note that most refractories made from silicates, upon being heated, decrease in density, in spite of the fact that the exterior volume may decrease. It is evident that this phenomenon is to be ascribed to molecular changes and to the progress of softening or fusion. In general, the porosity-temperature curves roughly parallel the specific gravity-temperature plots. Fire clays high in fluxing impurities invariably show a lower specific gravity than purer materials after firing to the same temperature, but even refractory clays exhibit an appreciable drop in specific gravity. Thus a high-grade Kentucky clay of specific gravity 2.60, after firing to 2820°F., possessed a specific gravity of 2.38. From the technical standpoint, the determination of the specific gravity in the case of silica brick is often of service in determining the degree of firing to which the brick has been subjected and in judging whether it has been brought to the proper end point.

The true specific gravity is also of great value in the theoretical studies for the observation of transformation points. From the exact standpoint, fusion represents a change of state which may be located by the discontinuity of some physical property. It may represent the intersection of the vapor-tension curves of the crystalline and amorphous states, the point of optical deorientation, the break in electrical conductivity, the discontinuity of the heating curve, the change in specific volume, etc. For this reason, plots showing the relation between specific gravity and temperature frequently establish critical points of considerable interest. At the same time, it may be entirely possible that even the most refined methods of the laboratory will fail in dealing with systems of great viscosity in which the high internal friction may arrest or greatly delay any changes by means of which the critical point is recognized.

Resistance to Compression at Furnace Temperatures.—In service, refractories are frequently subjected to compression and sometimes to tensile or shearing stresses, particularly in the arches and crowns of furnaces and kilns, gas-retort benches, hot-blast stoves, etc. Any decrease in rigidity suffered by the refractory body, due to incipient softening, will at once become evident by more or less marked deformation under stresses exceedingly small compared with the compressive strengths of the material in the cold state. Thus, a firebrick showing a cold crushing strength of 3000 lb. per sq. in. may be deformed under a pressure of 25 lb. per sq. in. at furnace temperatures. The softening of such substances through a given viscosity range is not only a function of temperature but of time as well. It is possible to vitrify a body by exposure to sufficiently high temperatures in a short time and to secure the same result at lower points of the temperature scale during longer periods.

Great differences exist in the viscosity and deformation of different materials, both natural and artificial. To generalize, it may be stated that the higher the impurities, i.e., basic oxides in the case of siliceous refractories and acid oxides in basic mixtures, the greater will be the deformation under conditions of pressure. Also, high-heat rick high in alumina usually show greater contraction under constant

pressure than do more siliceous ones. The resistance of refractories to pressure at furnace temperatures therefore offers a measure of their rigidity and, indeed, of their resistance to heat, since it is evident that a material showing but little deformation at a given temperature, other things being equal, is preferable to one that softens and shows excessive subsidence.

When considering resistance to load conditions, it should be remembered that the refractoriness of a material is a function of the pressure applied. For instance, a firebrick, showing a P.C.E. corresponding to cone 32 to 33, may under pressure show a steady deformation or even collapse at a temperature represented by cone 19 to 20. The initial mechanical strength and the degree of firing are factors in determining the resistance to load, since dense brick formed under a heavy pressure are, in general, more resistant than those of a more open and porous structure. The type and amount of bonding material present are involved as well. Almost invariably hard-fired brick will stand up much better under load than those burned at a lower temperature. It must be remembered, however, that the hot face of the refractory in a furnace wall is the only part that reaches or even approaches the true furnace temperature. Therefore, resistance to pressure at furnace temperature is by no means a simple property, and a test involving such a condition should not be prescribed indiscriminately. The test should be reserved for refractories that are to be used in places where pressure is really an important factor. The test as applied so far in this country errs on the safe side, but is likely to be unfair to certain materials.

Failure under load may occur in one of two ways, either through the softening of the mass, as evidenced by excessive contraction, distortion, or bending, or through shear, in which case the failure is usually more or less abrupt. The former type of failure is more common with fire-clay refractories and the latter with silica, chrome, and magnesite refractories.

The load test is fully covered by A.S.T.M. Designation C 16.

Although not a part of the standard method, the expansion and contraction of a brick in the load test may be measured by attaching a pantograph recorder to the beam. However, the specimen is rated according to the subsidence it has suffered, expressed in percentage of the original length, unless it has failed by collapse.

Schedule I is usually used for intermediate-heat duty brick, schedule II for high-heat duty, and schedule III for super-duty fire-clay brick. The test is no longer used for silica brick since the load that can be applied produces no subsidence in them at any of the standard test temperatures.

Thermal Properties.—The thermal properties of refractories, specific heat, conductivity, and expansion are determined according to the established physical methods. While these properties are of no little practical importance, the data available are meager and not very accurate.¹ The structure of the manufactured product, irrespective of its composition, is of paramount importance. Furthermore, these properties are subject to change with temperature, and comparatively few constants are available to establish accurately the character of these relations. Also, it must be realized that the structure of all these materials is certain to undergo physical changes that affect the thermal qualities. It is known that, in general, the specific heat and the thermal conductivity increase with temperature, but the fundamental laws governing these changes have not been established. The thermal conductivity of magnesite and silicon-carbide brick decreases with increasing temperature. They are, thus, exceptions to the general rule.

Specific Heat.—This constant is determined by heating a specimen of known weight in a furnace and, when it has been brought to constant temperature, allowing

¹ "Modern Refractory Practice," p. 212; WILKES, *Jour. Am. Ceram. Soc.*, Vol. 16, pp. 125-130, 1933; AUSTIN, *Jour. Am. Ceram. Soc.*, Vol. 17, pp. 173-177, 1934.

it to drop into a calorimeter. The error due to the heat lost in transferring the specimen is usually greater than has been appreciated. The evidence indicates that the increase in specific heat with temperature is, in general, a linear function, although this has not been established beyond doubt, and in some cases it is known that the relation is a more complicated one.

Table 1 from "Modern Refractory Practice," reprinted by permission of Harbison-Walker Refractories Co., gives the values for specific heat which are considered most probable:

TABLE 1.—SPECIFIC HEATS OF REFRACTORY MATERIALS
Mean Specific Heat between 32°F. and t °F.

t °F.	Fire-clay brick	Silica brick	Magnesite brick	Chrome brick
32	0.193	0.169	0.208	0.170
200	0.199	0.188	0.219	0.175
400	0.205	0.211	0.232	0.182
600	0.212	0.229	0.243	0.188
800	0.220	0.238	0.250	0.194
1000	0.227	0.246	0.257	0.199
1200	0.234	0.252	0.263	0.204
1400	0.242	0.256	0.267	0.208
1600	0.248	0.260	0.273	0.212
1800	0.253	0.264	0.277	0.216
2000	0.258	0.269	0.282	0.220
2200	0.261	0.272	0.287	0.222
2400	0.266	0.276	0.293	0.224

Thermal Conductivity.—The determination of the thermal conductivity of refractory materials is a matter of considerable difficulty. Tests carried out by different investigators have failed to agree, sometimes by as much as 50 per cent. In 1928, Committee C-8 of the A.S.T.M. appointed a subcommittee to study methods of test. The first report of this subcommittee presented tests made at five different laboratories. The results showed a wide variation in data on specimens taken from a carefully selected sample. The committee succeeded in interesting the Bureau of Standards in the problem of developing a suitable method, and the studies were undertaken in 1934. No suitable method had been developed up to the time it became necessary to set the work aside for the duration of the war. It is reported that considerable progress had been made. In the meantime, the subcommittee has continued its work on the data on insulating firebrick, from various laboratories, which are now in reasonably close agreement. At the recent meeting of Committee C-8, it was agreed to adopt a method depending on the water calorimeter method originally described by Wilkes.¹

It is recognized that this method has some shortcomings and probably does not give absolute values of thermal conductivity, but the committee feels that a method which gives comparative values, even though limited to insulating firebrick, is better than none. Check values within 5 to 10 per cent have been obtained with the apparatus at different laboratories.

¹ NORTON, *Jour. Am. Ceram. Soc.*, Vol. 25, pp. 451-459, 1942; WILKES, *Jour. Am. Ceram. Soc.*, Vol. 16, pp. 125-130, 1933.

It should be remembered that heat is transferred not only by conduction, but also by radiation and by convection. All three, conduction, radiation, and convection, play a part in the transfer of heat through a furnace wall. Refractory materials are quite porous in comparison with vitreous bodies, and heat is radiated from one cell wall to another, and minute convection currents are set up inside the hollow cells and in open joints. The brick in a furnace wall under gas pressure from

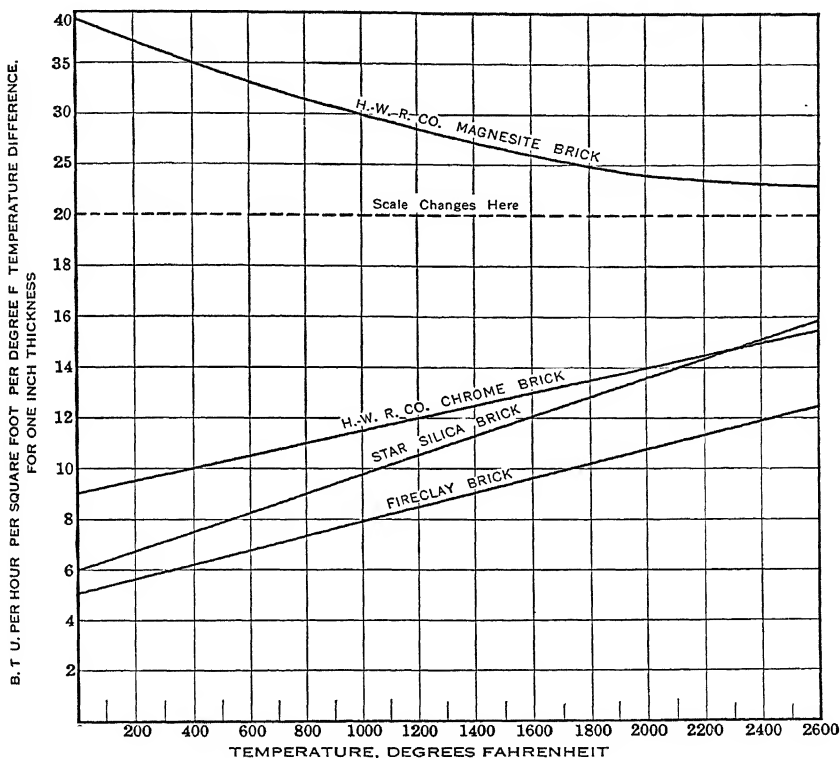


FIG. 4.—Thermal conductivity of refractories.

inside to outside show much higher temperatures than when the gas pressure is higher outside than inside the furnace chamber. Even in the conductivity apparatus we are really measuring heat transfer, in which all three types of heat transfer play some part, rather than measuring true conductivity. Only on a vitreous or refractory body of zero porosity could conductivity alone be involved. These facts must be borne in mind in using conductivity data for calculating heat losses through furnace walls. Figure 4 and Table 2, from "Modern Refractory Practice," give relative data.

Quoting in part from W. A. Hull,¹ " . . . it must be taken into consideration that only rough estimates of the heat transfer through the walls of a given installation

¹ HULL, *Chem. & Met. Eng.*, Sept. 13, 1922.

TABLE 2.—APPROXIMATE THERMAL CONDUCTIVITIES OF REFRACTORIES
B.t.u. per Hr. per Sq. Ft. of Wall Area, per °F. Temperature Difference, for 1 In.
Thickness

Kind of brick	Temperature, °F.													
	0	200	400	600	800	1000	1200	1400	1600	1800	2000	2200	2400	2600
Magnesite brick.....	39.5	37.2	35.4	33.3	31.5	30.0	28.3	26.9	25.9	25.0	24.0	23.5	23.2	23.0
Chrome brick.....	9.0	9.5	10.0	10.5	11.0	11.5	12.0	12.5	13.0	13.5	14.0	14.5	15.0	15.5
Star silica brick.....	6.0	6.8	7.5	8.3	9.0	9.8	10.5	11.3	12.0	12.8	13.5	14.3	15.0	15.8
Woodland fire-clay brick....	5.0	5.6	6.1	6.7	7.3	7.8	8.4	9.0	9.6	10.1	10.7	11.3	11.8	12.4

could be obtained by computation, in most cases, even though precise figures for the thermal conductivity of the firebrick were available. In the first place, the temperatures of the inner and outer surface must be known and, if the wall is to be more than one brick thick, the effect of the joint, or joints, must also be known as well as the conductivity of the refractory material, before the heat flow through the wall can be computed. Unless the temperatures of the two surfaces can actually be measured, a rough approximation is introduced at this point. The effect of joints must not be overlooked and introduces another factor that is uncertain as to magnitude.

"Furthermore, it must be taken into consideration that heat conductivity, or its reciprocal, resistance to heat flow, is but one of several factors working in series and that the larger the other factors, such as surface resistance and the effects of joints, the smaller, relatively, is the effect of differences or errors in the values assumed for thermal conductivity. The effect of surface resistance is shown in the operation of retorts in a gas plant, by the fact that when the flame changes from transparent to luminous the rate of gas production increases, showing that heat is passing into the retorts more rapidly. When the flame is transparent, a very large portion of the heat must be taken up by the retort by conduction; when the flame becomes luminous, radiant energy is transferred rapidly from the myriad of minute particles of incandescent carbon floating in the gas to the surface of the retort, and the retort is heated by radiation, reducing, to a large extent, the effect of the surface resistance. The fact that the difference in rate of production is considerable shows that the surface resistance is a large factor. This factor has been studied sufficiently to justify the statement that the surface resistance of one surface of a retort may be as great as the internal resistance of the retort wall itself.

"There is one other factor that should not be lost sight of in this connection. In the case of furnaces operating at high temperatures, there is a tendency for a portion of the brick to vitrify in the course of time, increasing the conductivity of that portion by possibly 40 to 50 per cent. Also, under certain conditions, refractories become impregnated with various substances taken up from slags or the gases penetrating them and it is practically certain that the impregnated materials differ to an important extent in heat conductivity and in other properties from the original material.

"It is not the purpose of the foregoing discussion to discourage the making of computations, but it is thought that some computers may believe, erroneously, that the lack of reliable figures for thermal conductivity is the greatest obstacle in the way of obtaining reliable estimates of the heat transfer to be expected in a given case, whereas, as a matter of fact, the more one studies the problem the less concern is felt about the accuracy of the value ascribed to the conductivity of the material in

question; other factors which necessarily enter into the computation have to be approximated so roughly that the uncertainty involved in this one becomes relatively unimportant."

Reversible Thermal Expansion.—The linear expansion upon heating specimens to known temperatures may be determined by direct measurements, using a micrometer microscope or Ames dial; by a system of levers, magnifying the expansion so it may be read from a scale; or by the interferometer method.¹

These measurements present relatively little difficulty between room temperature and 1800°F. since fused-quartz rods or disks may be used up to this temperature. As

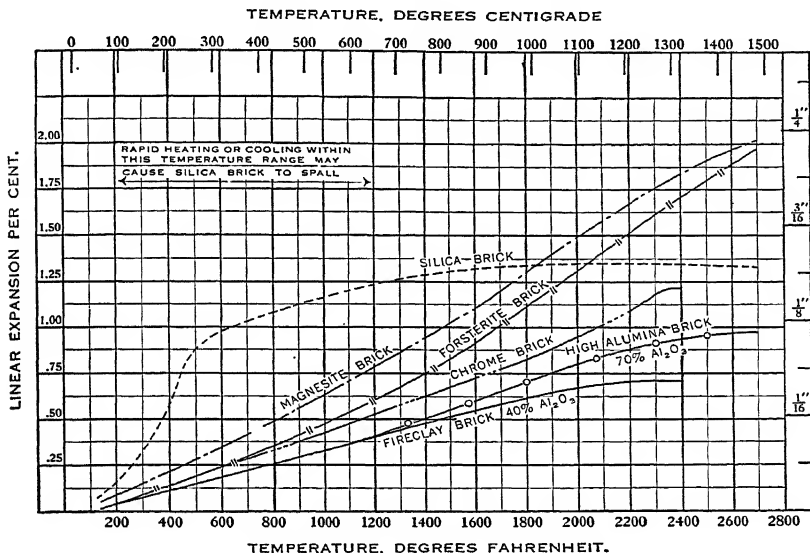


FIG. 5.—Thermal expansion of refractories.

the temperature is raised, the difficulties of accurate measurement increase, but the expansion can be measured with fair accuracy up to 2900°F. by the method described by R. A. Heindl.² Table 3 and Fig. 5 taken from "Modern Refractory Practice" give probable values of reversible thermal expansion.

Resistance to Temperature Changes, Thermal Spalling.—It is probable that no single property of refractory brick has been studied more intensively and persistently than that of resistance to spalling. Committee C-8 of the A.S.T.M. in 1921 adopted a water-dip test. This method was found to give unreliable results, and a modification was introduced which prescribed preheating the brick to a standard temperature. The brick were then subjected to the alternate heating and cooling, with one end first heated by being placed in the door of a hot furnace and then chilled by dipping in cold running water. This method, too, was found inconsistent with results in service and was replaced by a test developed by the Refractories Fellowship at Mellon Insti-

¹ MERRITT, The Interference Method of Measuring Thermal Expansion, *Jour. Research Nat. Bur. Standards, Research Paper* 515, Vol. 10, January, 1933.

² HEINDL, The Thermal Expansion of Refractories to 1800°C., *Jour. Research Nat. Bur. Standards, Research Paper* 562, Vol. 10, June, 1933.

TABLE 3.—APPROXIMATE REVERSIBLE THERMAL EXPANSION OF REFRACTORIES
Linear Per Cent—Mean Values

Tem- pera- ture, °C.	Kind of brick						Tem- pera- ture, °F.	Kind of brick					
	Silica	Fire clay 40% Al ₂ O ₃	High- alumina 70% Al ₂ O ₃	Magne- site and Magnex	Chrome and Chromex	For- sterite		Silica	Fire clay 40% Al ₂ O ₃	High alumina 70% Al ₂ O ₃	Magne- site and Magnex	Chrome and Chromex	For- sterite
100	0.14	0.05	0.05	0.10	0.06	0.05	200	0.13	0.05	0.05	0.09	0.05	0.05
200	0.55	0.11	0.10	0.21	0.14	0.12	400	0.58	0.12	0.10	0.22	0.15	0.12
300	0.95	0.18	0.16	0.33	0.23	0.22	600	0.96	0.19	0.17	0.35	0.24	0.23
400	1.05	0.24	0.22	0.45	0.31	0.32	800	1.07	0.25	0.24	0.48	0.34	0.35
500	1.13	0.31	0.29	0.58	0.40	0.43	1000	1.15	0.33	0.33	0.63	0.43	0.48
600	1.19	0.37	0.36	0.71	0.48	0.55	1200	1.22	0.40	0.40	0.78	0.53	0.61
700	1.24	0.44	0.44	0.86	0.57	0.67	1400	1.26	0.48	0.50	0.95	0.62	0.75
800	1.28	0.50	0.53	1.01	0.66	0.81	1600	1.30	0.55	0.60	1.12	0.72	0.92
900	1.31	0.57	0.63	1.18	0.75	0.97	1800	1.32	0.61	0.71	1.31	0.82	1.11
1000	1.32	0.61	0.73	1.34	0.84	1.14	2000	1.33	0.65	0.80	1.49	0.93	1.31
1100	1.33	0.65	0.81	1.50	0.94	1.32	2200	1.33	0.68	0.87	1.68	1.08	1.51
1200	1.33	0.68	0.87	1.67	1.07	1.50	2400	1.33	0.68	0.92	1.83	1.21	1.70
1300	1.33	0.69	0.92	1.80	1.21	1.68	2500	1.33
1400	1.32	0.96	1.94	1.85	2600	1.32	0.96	1.97	1.89
1500	1.30	0.98	2.05	2.00	2700	1.30	0.97	2.03	1.98

NOTE: To convert linear per cent expansion into thirty-seconds of an inch per foot, multiply by 3.8. For example, the linear expansion of silica brick at 2400°F. is 1.33 per cent. 1.33 multiplied by 3.8 equals 5.05, or approximately $\frac{5}{16}$ in. per ft.

tute. This test, with several minor modifications, is the standard today. A.S.T.M. Method C 38, which is here reproduced through the courtesy of the A.S.T.M., gives a general description of the test. There are also two specific standard tests: C 107, dealing with high-heat duty fire-clay brick, and C 122, dealing with super-duty fire-clay brick.

The Navy Department of the United States government uses a modification of the panel spalling test for tests and specifications, although the Bureau of Standards still uses a modification of the old water-dip test.

The standard A.S.T.M. test is applicable to fire clay and super-duty fire-clay brick only. Silica, magnesite, and chrome brick are more susceptible to temperature change than fire-clay brick. Attempts have been made to use the panel spalling test for these brick also, but so far no standard procedure has been adopted.

Although, in general, it may be said that the ability of refractories to resist sudden temperature changes is an inverse function of the coefficient of thermal expansion, many other factors affect this property. Furthermore, the physical changes brought about by the continued effect of heat in service may bring about a gradual change in the character of the refractory. Thus, a firebrick may resist sudden temperature changes very well when first placed in service but gradually become denser at the surface, through vitrification or the absorption of slag or glass, and thus acquire a decided tendency to spall. With some kinds of refractories, their tendency to spall when rapidly heated or cooled constitutes a limiting factor in their application in service.

Panel Spalling Test (A.S.T.M. Method C 38).

1. (a) This method of test outlines a general procedure for determining the resistance of refractory brick to the separate and combined effects of structural and thermal spalling. The test is believed to give a sufficient simulation of service conditions to suggest its use as a suitability test for certain classes of service that have been correlated with the test data.

NOTE.—Such correlation is now available in the case of super-duty and high-heat-duty fire-clay brick for stationary boilers, malleable iron furnace bungs, and incinerator furnaces.

(b) Detailed methods of test to be used in conjunction with this procedure in testing specific types of refractories are as follows:

Standard Method of Panel Test for Resistance to Thermal and Structural Spalling of High Heat-duty Fire-clay Brick (A.S.T.M. Designation C 107), and

Standard Method of Panel Test for Resistance to Thermal and Structural Spalling of Super-duty Fire-clay Brick (A.S.T.M. Designation C 122) of the American Society for Testing Materials.

Apparatus

2. The apparatus and equipment¹ shall consist of the following:

(a) *Panel Frame*.²—A panel framework shall be used which is essentially the same as that shown in Fig. 6.

(b) *Preheating Furnace*.²—A preheating furnace shall be used, constructed essentially like the design shown in Figs. 7 and 8.

(c) *Spalling Furnace*.—A spalling furnace shall be used, constructed in a manner as shown in Figs. 9, 10, and 11.

¹ The complete set of drawings necessary for the construction of the panel spalling equipment would require too much space to be included with the procedure, but these are available at a nominal charge from the Refractories Fellowship, Mellon Institute of Industrial Research, Pittsburgh, Pa.

² See figures, p. 345.

(d) *Temperature Measuring Instruments.*—The temperature of the test panel during the preheating shall be measured by means of an optical pyrometer. The spalling furnace temperature shall be determined by means of a platinum-platinum-rhodium thermocouple.

(e) *Air Supply.*—Means shall be provided for delivering air through each cooling unit at a specified rate.

Transfer Facilities.—A suitable means, such as a track or trolley, shall be provided for transferring the test panels.

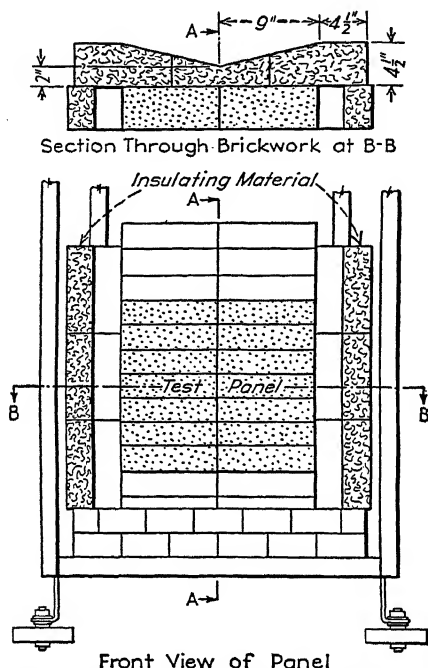


FIG. 6.—Panel frame and brickwork. The brick around the test panel are dummy brick which are used to help maintain uniform conditions within the panel.

Test Specimens

3. Whenever possible, the specimens for the test panel shall consist of 9-in. straight brick. When shapes of other sizes are to be tested, the specimens shall be cut so as to be not more than 3 in. in thickness by not more than 9 in. in length by not more than $4\frac{1}{2}$ in. in width.

Panel Construction

4. The specimens shall be laid up so as to form a test panel not less than 18 in. square, so that a $9 \times 2\frac{1}{2}$ -in. surface of each specimen will be exposed to the heat-treatment. Each specimen shall be labeled with ceramic paint on the face opposite that to be tested, and then weighed to the nearest 0.1 lb. There shall be placed about

the test panel a suitable grade of refractory dummy brick, in the manner shown in Fig. 6. All brick in the test panel shall be laid up with refractory kaolin unless otherwise specified. The joints shall be not thicker than $\frac{1}{16}$ in. The test panel, including the dummy brick, shall be insulated in accordance with the arrangement and design shown in Fig. 6.

Preheating Test Panels

5. The prepared test panels shall be placed in position on either side of the preheating furnace. When only one lot of brick is to be tested, a dummy panel should be used. The joint between the ends of the panel dummy brick and the preheating

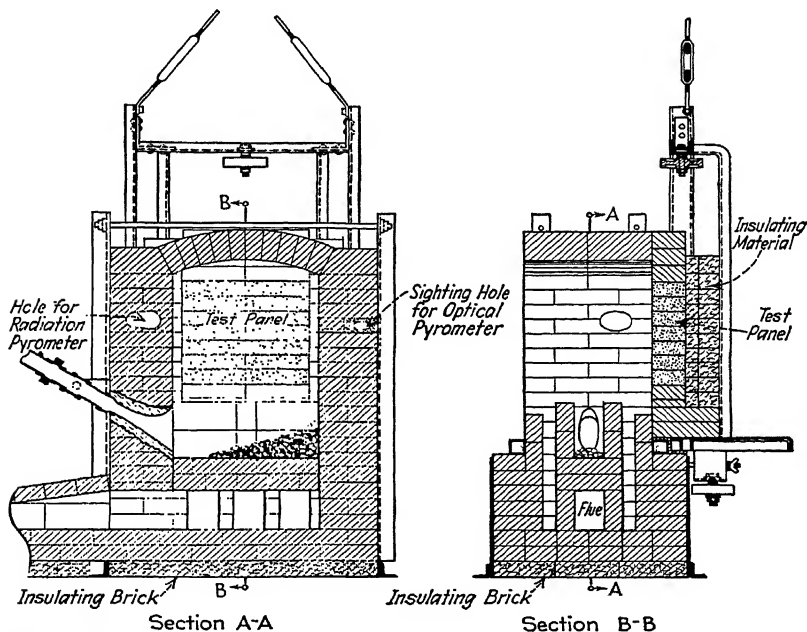


FIG. 7.—Design of preheating furnace. In Section B-B there is shown in position one of the two panels used during the preheating.

furnace shall be sealed with plastic kaolin. During the heating of the panels, the surface of the test specimen opposite that being heated shall be insulated in the specified manner so as to maintain a suitable temperature gradient in the specimen. The furnace shall be so operated that the temperature specified for preheating shall be reached in not less than 5 nor more than 8 hr. and maintained 24 hr. The permissible variation in temperature may be plus 20°C. and minus 25°C., but the average of the temperature during the run shall be that specified. After completing the preheating, the test panel shall be left in position for not less than 8 hr. before being removed. As a result of this heat-treatment, any tendency toward fusion, vitrification, swelling, or shrinkage will be evident and any development of structural spalling will be apparent. A record shall be made of the condition of the brick specimens, indicating the extent

of the fusion, vitrification, swelling, or shrinkage. If shrinkage takes place a record shall be made of its nature, *i.e.*, whether or not this occurs as a shrinkage of each brick as a unit, or as the development of cracks on the surface of each brick. The surface of the test panels may be photographed.

Procedure

6. The test panels shall then be subjected to thermal spalling by heating them within a specified time to a specified temperature and rapidly cooling by means of a prescribed blast. The panels shall be placed in position in front of the spalling furnace

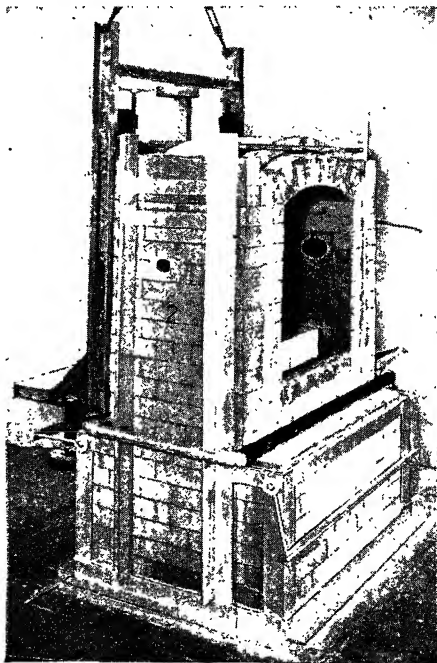


Fig. 8.—Preheating furnace with panel in position on far side of furnace.

in the manner shown in Figs. 10 and 11. The temperature of the spalling furnace shall be raised within 3 hr. to the specified temperature. During the heating period, the panels shall be alternately placed in position in front of the furnace so that each may be heated to about the same temperature. The position of the panels shall be changed at temperatures of 400, 200, and 100°C. below the final furnace temperature. During this period, the test panels shall not be cooled by means of the cooling blast. After the final temperature of spalling is obtained, the panels shall be held in position for a specified time, and then shifted so as to expose the heated panel for cooling. After each panel has been subjected to the required number of cycles, the furnace shall be allowed to cool, but the cooling procedure for the test panels shall be conducted so that each panel receives two additional cycles without the water mist.

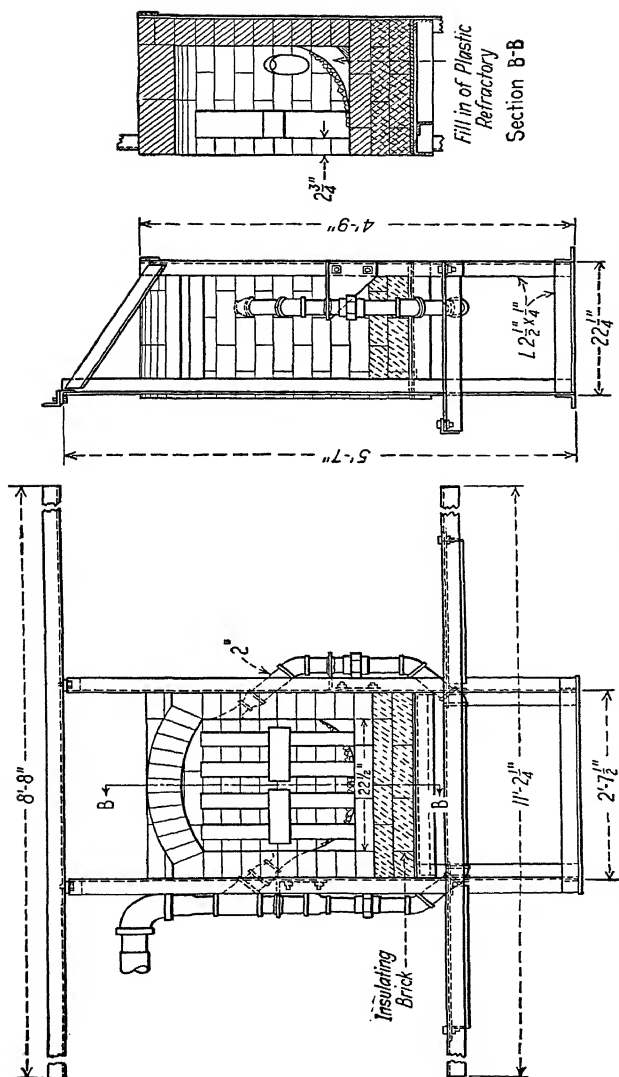


Fig. 9.—Furnace of the spalling equipment. The complete layout is shown in Figs. 10 and 11.

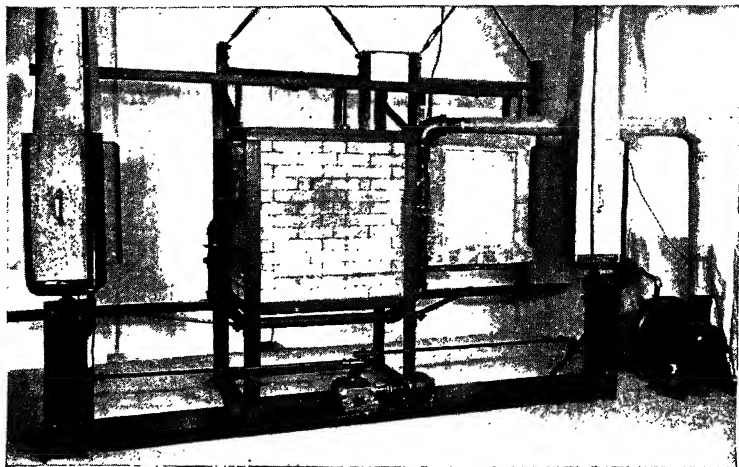


FIG. 10.—Cooling equipment for panel spalling test. View from the back of the furnace showing the panels in position, the nature of the mechanism which causes the air manifolds to operate with a reciprocating motion, and the motor blower unit used to operate the furnace.

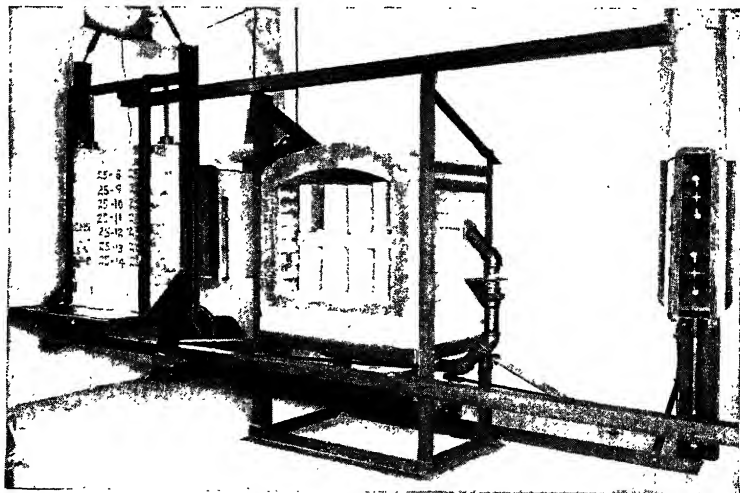


FIG. 11.—Spalling equipment for panel spalling test. Front view showing one of the two panels in position. During the test a second panel is placed in front of the furnace opening. The two panels are linked together so that when the one which is now in the position for cooling is moved in front of the furnace, the other panel will be in position in front of the right-hand cooling unit. By this means, each panel is alternately heated and cooled.

Dismantling Test Panel

7. When the test panel has cooled sufficiently so that the brick can be handled, the panel shall be carefully dismantled. Each of the test specimens shall be cleaned to free it of kaolin and in such a manner as to remove spalls, after which the specimens shall be reweighed to the nearest 0.1 lb., and the loss in weight recorded as percentage of the original weight. When a second heating treatment is given and the 9-in. brick specimens have been numbered from 1 to 7 from top to bottom in the test panel, their



FIG. 12.—Arrangement of test brick for making photographic report. The brick are in the same corresponding position as laid up in the panel. The percentage loss and photograph number or designation are also shown.

order shall be changed so as to assume the positions of 4-3-2-1-7-6-5. In the case of the six 3-in. brick specimens the order shall be changed so as to assume the positions of 3-2-1-6-5-4 for the second heating treatment. The specimens shall also be turned end for end.

Report

8. The report may include a photograph of the test panel after preheating, and shall include comments as to its condition. The average spalling loss for the whole

panel shall be given, as well as a photograph of the spalled brick specimens arranged in the manner shown in Fig. 12.

Resistance to Slagging Action.—Although the effect of slag on refractories is of primary importance in determining their life in service, no quantitative test has yet been found which adequately correlates test data and actual performance. The absorption of molten material by a refractory depends on the structure and chemical composition of the refractory, the temperature, and the viscosity and chemical composition of the slag or glass. It is evident that for highest slag resistance the refractory should be dense and as impervious as possible and not subject to chemical interaction when in contact with the liquid slag. It appears, however, that not only the total volume of pore space but also its capillary character, *i.e.*, the fineness and continuity of the pores, is of great significance in this respect. Sometimes a slag will act as a glaze over the surface of the furnace wall and the action be merely a slow, even washing away of the refractory, but if the temperature is raised even a little, the glaze vanishes, the slags penetrate the brick by capillary action, and the brick is quickly honeycombed. It is sometimes a question as to which is more important, a dense brick that retards slag penetration or one that is of such a chemical nature that the slag is slow to react. A dense brick of the correct chemical composition can seldom be found.

Some years ago the A.S.T.M. adopted a tentative slag test in which a ring $2\frac{1}{2}$ in. inside diameter and $\frac{1}{2}$ in. deep was cemented to the surface of the test brick, or a similar cup cut into the brick. The ring or cup was then filled with ground slag and the brick heated to a specified temperature, usually 2462°F. After the test the brick was sawed through the center of the ring and the depth and area of the slag penetration measured. This test was abandoned because it gave more emphasis to the permeability of the brick to the slag than to its tendency to react. The test is still used as a research tool but must be interpreted with caution. A test in which the slag is blown into a pot furnace from a rotating burner has been developed by R. K. Hursh¹ at the University of Illinois. This test has some features to commend it but is still far from standardized. Reviews of the literature on slag tests have been published from time to time, the most recent being by T. L. Hurst and E. B. Read.² It appears probable that no single slag test that will cover all conditions can be developed. It may be necessary to use several tests, each covering a definite condition.

Mechanical Strength.—The mechanical strength of refractories in the cold state varies between wide limits, according to the density and hardness of the materials. Thus, fire-clay brick may show compressive strengths from 1000 to 5000 lb. and moduli of rupture up to 1200 lb. per sq. in. The toughness of refractories, as measured by their resistance to abrasion, likewise varies greatly. Many refractories that resist very high temperatures stand up but poorly under conditions of abrasion. Under certain conditions, as in the top part of iron blast furnaces, hard and tough materials are required to resist the wearing action of the charge, while for other uses this requirement may not be essential.

The toughness of refractory bricks may be determined by means of the "rattler," a machine employed for the testing of paving brick, with the modification that the entire charge consist of the smaller spheres employed in the standard test of the A.S.T.M., C 7, or by running the test for 600 revolutions instead of 1800, or by leaving the entire charge of balls out of the barrel and tumbling the bricks upon themselves. This test has not been standardized. At least two other abrasion tests have been proposed, one by Hartmann and Hougan, Hot and Cold Abrasion Tests, American

¹ HURSH, A Laboratory Slagging Furnace with a Rotating Burner; *Jour. Am. Ceram. Soc.*, Vol. 22, pp. 354-357, 1939.

² HURST and READ, *Jour. Am. Ceram. Soc.*, Vol. 25, pp. 283-294, 1942.

Electrochemical Society, 1920; a second by Harvey and McGee.¹ Neither has found any place in testing procedures.

Committee C-8 endeavored to develop a hot abrasion test and a method was proposed by J. B. Shaw.² This method, too, fell by the wayside.

The mechanical strength of refractories while hot has been studied by McGee,³ Cole,⁴ Hunt and Bradley,⁵ and Hartman and Koehler.⁶

Types of Refractories.—The several types of refractories may be classified from the technical standpoint as follows:

A. Refractories of the silica-alumina series.

1. Fire-clay refractories.
2. Silica refractories.
3. High-alumina refractories.
 - a. Diaspore type.
 - b. Other types.

B. Basic refractories.

1. Magnesite refractories.
2. Chrome refractories.
3. Dolomite refractories.

C. Special refractories.

D. Insulating refractories.

REFRACTORIES OF THE SILICA-ALUMINA SERIES⁷

The decade just past has witnessed a remarkable change in the character and types of refractories available for furnace construction. Not much more than fifteen years ago the production of refractories of the alumina-silica series consisted almost solely of silica and fire-clay brick. Silica-brick linings came at one end of the series with about 96 per cent silica and 1 per cent alumina, the other 3 per cent being made up of lime and other oxides. Next came siliceous fire-clay brick with about 76 to 85 per cent silica and 21 to 12 per cent alumina. Second-quality and first-quality fire-clay brick, as they were called at that time, filled in the range from 62 per cent silica and 31 per cent alumina to 53 per cent silica and 43 per cent alumina. A few high-alumina brick were made from Missouri diaspore, or from bauxite and fire clay, and brick were also available, although little used, which were made from electrically fused alumina of a high degree of purity. Today, brick covering practically the entire range of composition from about 98 per cent silica to 98 per cent alumina, except for an interval from about 1 to 12 per cent alumina, are finding commercial application. Even in this short interval, silica mortars are found. In the high-alumina range, the industry has standardized on diaspore brick compositions of 50, 60, and 70 per cent alumina. There are also available brick of 80 and 90 per cent alumina and, finally, practically pure electrically fused Al_2O_3 brick. These refractories fill out the gap between 42 and, say, 98 per cent alumina which was formerly but sparsely covered.

¹ HARVEY and MCGEE, Factors Affecting the Resistance of Silica Refractories to Abrasion, *Jour. Am. Ceram. Soc.*, Vol. 7, p. 895, 1924.

² SHAW, *Jour. Am. Ceram. Soc.*, Vol. 13, pp. 427-436, 1930.

³ MCGEE, Comparison of Hot and Cold Modulus of Rupture for Silica Brick, *Jour. Am. Ceram. Soc.*, Vol. 5, pp. 888-900, 1922.

⁴ COLE, Relation of Crushing Strength of Silica Brick at Various Temperatures to Other Physical Properties, *Jour. Am. Ceram. Soc.*, Vol. 15, pp. 611-621, 1932.

⁵ HUNT and BRADLEY, *Jour. Am. Ceram. Soc.*, Vol. 20, pp. 267-269, 1941.

⁶ HARTMAN and KOEHLER, Physical Characteristics of Specialized Refractories, Part IV, Cross Breaking Strength, *Trans. Am. Electrochem. Soc.*, 1921.

⁷ The data on the alumina-silica series are largely taken from a paper by Fred A. Harvey presented before The Electrochemical Society and published in Vol. 75 of the society's transactions.

The development of refractories in this field has included not merely the manufacture of diaspore brick of standard high-alumina composition, but has utilized materials other than diaspore as the source of alumina and has introduced varying methods of manufacture. Of particular importance have been the changes that have taken place in manufacturing methods. Formerly, brick were made mainly by hand or by the auger machine, and little diversity, except in raw materials, existed. Today it is possible to grind and screen portions of the brick batch separately and then to recombine them under high pressure in the manner that experiments have shown to be most advantageous and, if desired, to remove the air from the ground clays by the application of a vacuum during pressing. The more important principles of grain

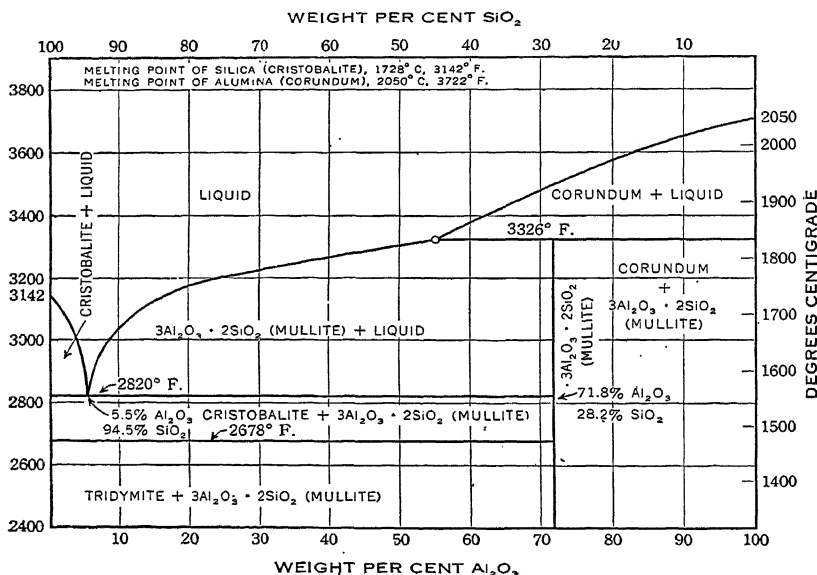


Fig. 13.—Alumina-silica equilibrium.

sizing have been established, and there have been evolved some particular products which have properties of outstanding value.

The variety of refractories is becoming increasingly large, and the selection of the refractory that will best be adapted to some particular service condition consequently requires expert knowledge.

The Alumina-Silica Equilibrium Diagram.—The equilibrium diagram for mixtures of pure alumina and pure silica is the logical starting point in any discussion of refractories of the alumina-silica series. It must be recognized that this diagram represents ideal conditions, involving only two components, and those under a condition in which equilibrium has been attained. Neither of these conditions is ever reached in the manufacture of refractories. The crude materials contain accessory oxides which, in general, lower the temperature at which liquids begin to form, and the refractory is usually in a state far from equilibrium. Nevertheless, much of the knowledge of the subject must be based on these fundamentals.

Figure 13¹ shows the equilibrium diagram for the alumina-silica system. The curved heavy line represents the temperatures at which any possible mixtures of pure alumina and pure silica become entirely liquid. Below this curved line the mixture may be entirely solid or may consist of solid and liquid. The various areas marked off on the diagram show the composition of the mixtures, for equilibrium condition. Disregarding accessory oxides, all the refractories of the alumina-silica series fall somewhere within the limits of Fig. 13. Silica brick are close to the extreme left and fused-alumina brick to the extreme right.

The Firing Behavior of Clays.—The firing behavior of clays has been touched upon on page 353, and a method of test by "draw trials" has been described.

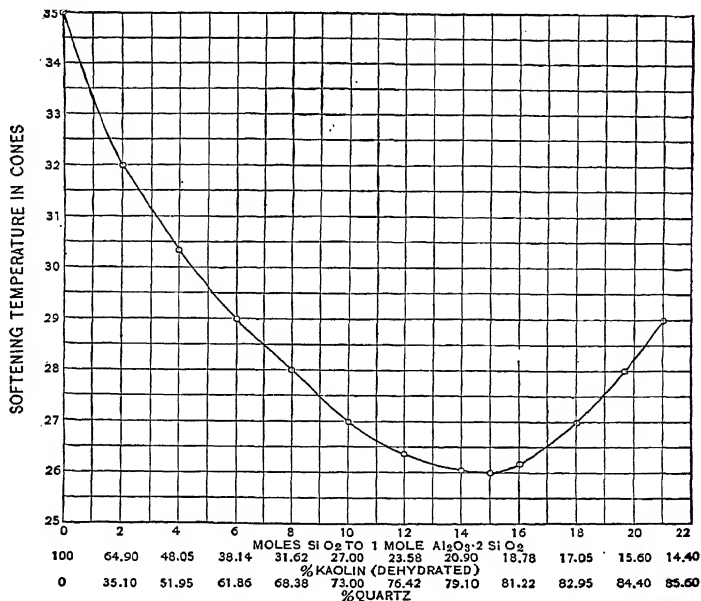


FIG. 14.—Seger's softening-temperature diagram.

In estimating the value of a bond clay, it is particularly desirable to study its characteristics at a series of temperatures. It is useful to know the volume and porosity changes, showing the temperature at which the clay becomes dense or vitrified, the range of vitrification, and the temperature at which the clay begins to overfire. With bond clays, it is usual to draw the first specimen at 1050°C. and from there on at each additional 25°C. until overfiring becomes pronounced. The rate of temperature rise of the furnace should not exceed 25°C. per hr. Other things being equal, a clay with a long range of vitrification, *i.e.*, the temperature interval between the vitrification and overfiring temperatures, will be more useful and easier to fire than one of short range. For some specific purposes, such as acid-resisting fire-clay brick, the minimum porosity reached is of great importance. Figure 2 shows the volume and porosity curves for a long and short-range clay. Tables 4 and 5 give data on physical tests of a number of refractory bond clays.

¹ BOWEN and GREIG; The System: $\text{Al}_2\text{O}_3\text{-SiO}_2$, *Jour. Am. Ceram. Soc.*, Vol. 7, pp. 238-243, 1924.

TABLE 4

Num- ber	Source of clay	Water of plasticity	Drying shrinkage	Ratio pores to volume of shrinkage water	Time of slaking 1 clay 1 sand, min.	Apparent specific gravity of dried clay	Modulus of rupture of 1 clay 1 sand, lb. per sq. in.	Temperature of vitrifi- cation, °C.	Overfir- ing tem- perature, °C.	P.C.E.
1	English ball clay.....	44.90	43.30	0.79	30	1.69	323	1100	1320	31½
2	Klingenberg, A. T.....	39.68	42.58	0.57	78	1.79	381	1125	1450	32
3	Klingenberg, E. T.....	50.66	55.05	0.56	108	1.70	363	1100	1450	32
4	Missouri.....	43.63	43.63	0.72	33	1.72	351	1175	1450	32
5	Kentucky.....	45.28	34.53	1.05	9	1.55	234	1260	1450	32
6	Ohio.....	25.30	27.92	0.83	9	1.94	309	1230	1350	30½
7	Ohio.....	22.08	24.46	0.80	8	2.01	281	1290	1400	31
8	Missouri.....	34.66	42.63	0.54	117	1.90	554	1175	1320	27
9	Illinois.....	45.16	45.35	0.65	54	1.67	341	1200	1450	32
10	Kentucky.....	50.85	46.80	0.71	37	1.57	362	1230	1450	32
11	Tennessee.....	44.73	35.27	0.97	27	1.56	282	1175	1450	32
12	Maryland.....	38.91	40.61	0.67	114	1.76	518	1175	1390	31
13	Illinois.....	40.98	38.81	0.75	45	1.66	262	1285	1475	32
14	Mississippi.....	31.39	31.36	0.81	55	1.81	326	1290	1450	30
15	English ball clay.....	40.33	39.85	0.75	41	1.71	389	1100	1425	32

Similar results for fire clays of the No. 2 and lower types are reported in Table 6.
SCHUBERT, H. G., *Jour. Amer. Ceram. Soc.*, Vol. 1, pp. 267-286.

TABLE 5

Num- ber	Source	Water of plasticity	Drying shrinkage	Apparent specific gravity of dried clay	Modulus of rupture of clay dried at 110°C.	Shrinkage at maximum firing contraction	Temperature of vitrification in cones	P.C.E.
1	Roseville, Ohio.....	27.5	30.8	2.08	218.6	17.12	14+	26+
2	Magadore, Ohio.....	19.1	16.1	2.02	142.5	14.10	14+	18
3	N. Brighton, Pa.....	21.0	20.0	1.99	194.2	12.43	14	28
4	Crooksville, Ohio.....	24.3	22.7	1.93	173.8	13.70	14+	28
5	Nelsonville, Ohio.....	23.0	22.7	1.9	325.0	13.30	14+	31

Fire-clay Refractories.—The P.C.E. of the purest clays is about cone 34, corresponding to about 3200°F., *i.e.*, approximately the melting point of pure platinum (3224°F., 1773.5°C.). Clays with a P.C.E. below about cone 19 (about 2768°F.) should not be classed as fire clays. The A.S.T.M. definition for fire clay is as follows: "A sedimentary clay of low flux content" (A.S.T.M. Designation C 71). Although the chemical composition of fire clays approaches more or less closely that of kaolinite, $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$, wide differences exist as regards their physical structure, varying through all stages from the well-defined crystalline state to that of a typical colloid. From the technical standpoint, roughly, four classes of refractory clays may be distinguished, *viz.*, kaolins, flint clays, semiflint clays, and plastic clays.

Kaolin.—The purest type of clay is that represented by carefully washed kaolin, consisting essentially of a hydrous silicate of aluminum, $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$, corresponding to the following percentage composition:

	Dry basis, per cent	Calcined basis, per cent
Silica.....	46.5	54.1
Alumina.....	39.5	45.9
Combined water.	14.0	

Ross and Kerr¹ state: "Kaolinite is the dominant mineral in residual kaolin. Kaolinite is almost the only kaolin mineral that forms extensive deposits and is the source of most if not all of the kaolin and 'china clay' of commerce."

Kaolinite was reported by Grimes and Bray² to be present at least to the extent of 40 per cent in flint clay from Phelps County, Mo., and in diaspore clay from the same location.

However it seems unsafe to assume that kaolinite is the fundamental constituent of all fire clays, since relatively few fire clays have been studied and reported.

While an extended discussion of the nature and classification of the clay minerals is not justified here, it seems well to mention that there are six known mineral constituents of the kaolin group. Kaolinite, nacrite, dickite, and halloysite, all have the same chemical composition, while anauxite has the composition $\text{Al}_2\text{O}_3 \cdot 3\text{SiO}_2 \cdot 2\text{H}_2\text{O}$ and allophane, $\text{Al}_2\text{O}_3 \cdot n\text{SiO}_2 \cdot n\text{H}_2\text{O}$. An excellent discussion is found in the Orton lecture by Kerr.³

Georgia kaolins are being used in the manufacture of both kaolin and fire-clay brick. The volume shrinkage of these kaolins is high, and this fact introduces considerable difficulty into its use for brickmaking. Were it not for their distance from the large consuming markets, these kaolins would be more widely used.

Flint Clays.—The composition of high-grade flint clays is close to that of the kaolins, even as to the content of chemically combined water. The flint clays are highly refractory and show a P.C.E. from cone $32\frac{1}{2}$ to 34. Physically, they are unlike the soft and chalky kaolins in possessing a hard, dense amorphous structure, showing a well-defined conchoidal fracture. The color is usually gray. They have little natural plasticity, and this property is not greatly improved by weathering. It is possible by

¹ ROSS and KERR, The Kaolin Minerals, *Professional Paper* 165, Shorter Contribution to General Geology, U.S. Geol. Survey, 1930.

² GRIMES and BRAY, The Mineral Constitution of Various Ceramic Clays, *Jour. Am. Ceram. Soc.*, Vol. 19, pp. 307-315, 1936.

³ KERR, A Decade of Research on the Nature of Clay, *Jour. Am. Ceram. Soc.*, 21, 267-286, 1938; Also, NORTON, "Refractories," 2d ed., pp. 130-150, McGraw-Hill Book Company, Inc. 1942.

suitable grinding treatment and the addition of some organic materials to develop sufficient plasticity for the molding of brick either by hand or on the power press. Flint clays occur extensively in Pennsylvania, Kentucky, and Missouri and to some extent in Maryland, Ohio, Colorado, Georgia, California, and Oregon. They differ markedly in their burning characteristics. Many of the Pennsylvania flint clays and some of the others show the characteristic known as "secondary expansion." When clay is fired, it first shrinks, then expands (secondary expansion). It may again show some shrinkage on reaching higher temperatures before the overfiring temperature is reached and attendant expansion starts. The grain size is of importance in this connection. A coarse grind will exhibit much greater secondary expansion than a fine grind, but a relatively coarse grind is necessary to the development of certain desirable properties of the brick for particular services.

It is not customary to make brick of flint clay alone, although this has sometimes been done. Usually the grains of flint clay are bonded with 15 to 30 per cent of semihard or plastic clay. It is customary also to incorporate a certain proportion of waste brick bats of the same composition as that of the brick being manufactured. This helps to decrease shrinkage and cracking in drying and burning and is most useful in the case of large and irregular shapes.

Semiflint Refractory Clay.—As is suggested by their name, the semiflint clays lie between the true flint and the plastic clays. They occur in all districts, although the name is most commonly used in Pennsylvania. As a rule, they have a P.C.E. between cone 31 and $32\frac{1}{2}$. They are subject to weathering and develop considerable plasticity and bonding strength. They are used with flint clay as bonding clays or as the main portion of the brick mix.

Plastic Refractory Clays.—Clays combining high plasticity with high refractoriness are not of common occurrence. While there are many plastic clay deposits in the United States, the majority of these materials show plasticity at the expense of heat-resisting power. Much of the plastic or bond clay used in the manufacture of refractories has a P.C.E. of cone 28 to 32, and occasionally clays as low as cone 18 are used in high-heat duty brick for special purposes. For intermediate and low-heat duty brick the mix usually consists of semihard and plastic or of plastic clay alone with or without grog (*i.e.*, bats). There is an abundance of plastic clay of this quality in Pennsylvania, Kentucky, and Ohio. In Missouri, there are extensive deposits of clay of a P.C.E. value of about cone 32 to $32\frac{1}{2}$, gray in color and of greater plasticity than the so-called flint clays of that state.

Important deposits of plastic clays are the so-called ball clays of Tennessee and Kentucky, and the plastic fireclays from St. Louis and southern Ohio. These clays are of great importance as sources of clay for the manufacture of refractories for the glass industry and of graphite crucibles. Plastic clays of good quality are also found in Colorado, Utah, and California.

Typical analyses of clays of the various types and districts are given in Table 6.

Siliceous Clays.—It has been customary to divide refractory clays into two types, one high and the other low in silica. The advantage of such a division arises from the fact that the two types behave quite differently in service. The division line between the two kinds of materials has been arbitrarily fixed at a silica content of 70 per cent, on a calcined basis. It is obvious that the gradation from one class to the other is not an abrupt one and that the properties of the first gradually merge into those of the second. These clays thus consist of clay minerals plus a considerable amount of free silica as quartz sand or chert. Some siliceous clays are also micaceous.

There are several characteristics by means of which the two types may be distinguished. As a class, although with certain exceptions, the siliceous clays possess less plasticity, bonding power, and strength in the dry state than do the more alumi-

nous clays, and also a lower drying shrinkage. In the firing process, they usually show no decided shrinkage and hence remain porous.

It was realized early in the study of fire clays that any addition of free silica to pure clay substance lowered the softening temperature. Thus Seger¹ found that a mixture corresponding to the molecular formula $\text{Al}_2\text{O}_3.17\text{SiO}_2$, composed of kaolin and ground quartz, softened at cone 26, while the clay itself fused at cone 35. This composition was believed to represent the eutectic between the two components, consisting of 23 per cent hydrous clay substance and 77 per cent quartz. Seger's eutectic composition varied considerably from the true point as determined by Bowen and Greig² (see Fig. 13). The curve of Seger is given in Fig. 14.

TABLE 6

	P.C.E.	SiO ₂	Al ₂ O ₃	TiO ₂	Fe ₂ O ₃	CaO	MgO	Na ₂ O	Ign. loss	Total
Pennsylvania:										
Flint clay.....	33-34	37.0	2.4	1.6	0.1	0.3	0.9	13.8	99.9	
Semiflint.....	32	32.3	2.0	1.7	0.2	0.5	2.6	10.2	99.8	
Plastic.....	30	27.2	1.5	2.8	0.2	0.5	3.0	9.4	100.6	
Nodule.....	34	53.0	2.8	5.3	0.3	0.2	1.6	14.5	100.1	
Siliceous.....	30-31	19.7	1.0	0.4	0.1	0.6	1.5	6.9	99.9	
Ganister.....		0.83	0.12	0.46	tr.	0.06	0.30	0.35	100.22	
Kentucky:										
Flint.....	33-34	36.8	2.2	1.5	0.1	0.3	0.9	13.5	99.8	
Semiflint.....	32-32½	31.8	2.5	1.5	0.2	0.6	1.5	11.4	100.2	
Plastic.....	30	27.0	1.7	2.2	0.3	1.0	2.6	8.8	99.9	
Ohio:										
Flint.....		34.1	2.1	2.4	0.2	0.4	1.3	12.9	99.8	
Plastic.....	30-31	26.5	1.8	2.0	0.4	0.7	1.8	9.6	100.2	
Missouri:										
Flint.....	33-34	38.5	2.2	0.8	0.2	0.4	0.8	13.6	99.9	
Semiflint.....	32½	34.1	1.1	1.6	0.4	0.4	1.7	11.4	100.1	
Plastic.....	31-32	27.7	1.6	2.4	0.4	0.3	0.4	12.4	99.8	
No. 1 diaspore.	38	.3	3.5	1.5	0.2	0.4	1.0	14.0	100.3	
Burley diaspore	36	55.6	3.1	1.4	0.1	0.4	2.0	13.1	100.2	
New Jersey:										
Siliceous.....	30	15.8	1.7	0.5	0.3	0.3	0.2	5.6	100.6	
Plastic.....	30	22.3	1.4	1.1	0.2	0.8	1.7	8.0	100.7	
Georgia:										
Kaolin.....	33-34	37.1	1.4	1.1	0.4	0.2	0.3	13.5	100.2	
Texas (Athens):										
Siliceous clay..	32-32½	26.8	1.7	1.2	0.8	0.5	0.5	9.9	99.8	

The Testing of Refractory Clays.—In the study of clays it soon becomes evident that the chemical composition is not the only criterion of their value. It is impossible to foretell anything concerning the physical properties of a clay from its composition, except the refractoriness, which can be estimated with fair accuracy. For this reason, tests of the physical properties are of great significance. It might be well to consider some of these, together with simple tests devised for their numerical evaluation.

¹ SEGER, "Collected Writings," Vol. 1, p. 434.

² BOWEN and GREIG, The System: $\text{Al}_2\text{O}_3.\text{SiO}_2$, *Jour. Am. Ceram. Soc.*, Vol. 7, pp. 238-254, 1924.

In testing clays to investigate their suitability for refractory purposes, the following method may be of use.

TABLE 7.—GROUP TESTS FOR CLAYS
To Investigate Their Suitability for Various Uses

HWRCO Method	Tests to be Run	Plastic clays	Semi- flint clays	Flint clays
14	P.C.E.	X	X	X
17	Chemical analysis, complete	X	X	X
11	Draw trial, h-p cylinders	X	X	X
	Draw trial, lumps	—	X	X
	Bulk specific gravity of lumps	X	X	X
29	Water of plasticity	X		
13	Ring test and working time on 20-mesh sample	X	X	
29	Linear drying shrinkage	X		
29	Dry modulus of rupture	X	X	
29	Bonding strength, dry	X	X	

The tests called for in Table 7 are mostly covered by A.S.T.M. Standard Tests or by Harbison-Walker Standard Method 29 which follows:

DRYING SHRINKAGE, WATER OF PLASTICITY, AND BONDING STRENGTH OF FIRE CLAYS

Apparatus and Materials.—A mold for making bars 1 in. square and 7 in. long. Standard Ottawa sand.

Grind of Sample.—Mortars and fine ground fire clays are to be tested as received. Crude materials are to be ground to pass 20-mesh Tyler hand screen.

Mixing with Water.—The ground clay or other material shall be mixed with water to a plastic consistency and then thoroughly kneaded by hand. The consistency shall be such that when the test piece is removed from the mold, it can be handled without appreciable bending. It should contain the maximum amount of water which will still permit this operation. This may be judged by making the plastic mass as wet as possible without its sticking to a steel spatula as it cuts through the mass.

Molding.—The mold shall be thin and evenly oiled with kerosene or a light machine oil only as frequently as is necessary to keep the clay from sticking. The test piece shall be formed by taking in the hand a lump of clay, somewhat larger than required to fill the mold, and kneading it into a roll approximately the length of the mold. It shall then be placed in the mold and forced into the corners by blows with the cheek of the fist. The excess shall then be struck off with a wire and the piece slicked with a spatula and appropriately marked or numbered for identification. Care should be taken by the operator to keep his hands free from oil. The test piece shall be removed from the mold immediately and transferred to a smooth, straight pallet, care being taken not to distort it.

Three test pieces of each mix are required for determinations of drying shrinkage and a minimum of 10 pieces for bonding strength.

Marking for Shrinkage.—With a sharp-edged spatula, a line shall be drawn lengthwise of the freshly molded specimen and at right angles to this line two cross marks will be placed $6\frac{3}{4}$ in. apart. (These marks will, therefore, be one hundred $\frac{1}{16}$ in.

apart and, in measuring the shrinkage in drying, $\frac{1}{16}$ in. will be equivalent to 1 per cent linear shrinkage.)

This method of marking is suggested as a convenient short cut; in any case the measurements are to be made between two parallel marks on the surface of the specimen and not between end faces.

Water of Plasticity.—One bar is to be weighed as soon as molded and again after drying. From these data, water of plasticity is calculated as follows:

$$\% \text{ water of plasticity} = \frac{\text{wet weight} - \text{dry weight}}{\text{dry weight}} \times 100$$

Drying.—The bars are to be allowed to dry for about 6 hr. in air at room temperature and then for 15 to 24 hr. at 120°C.

Drying Shrinkage.—Upon completing the drying treatment, the distance between the parallel marks is to be measured. Linear drying shrinkage is then calculated from the following formula:

$$\text{Linear shrinkage} = \frac{\text{wet length} - \text{dry length}}{\text{wet length}} \times 100$$

Bonding Strength.—Modulus of rupture (average 10 minimum) shall be considered a measure of bonding strength of the raw clays. It shall be obtained on bars molded and dried in accordance with the general method (above) on test bars of the following mixes:

1. 100 per cent clay.
2. 50 per cent clay.
50 per cent standard Ottawa sand.

NOTE:—Portions of this method were adapted from the 1928 American Ceramic Society Standard; *Jour. Am. Ceram. Soc.*, Vol. 2, p. 450, 1928.

End of Harbison-Walker Method 29.

A purely arbitrary method for the determination of plasticity of refractory mortars and plastic clays has been in use at the Harbison-Walker Laboratory for several years. It represents no striking advance in knowledge, but because of its usefulness and the lack of a better method, it is reproduced here.

PLASTICITY TESTS FOR REFRACTORY MORTARS

Scope: (1) This method covers the procedure for determining the following properties of refractory mortars:

- A. Water retention (by ring test).
- B. Spreading time and working qualities.
- C. Rate of settling, normal procedure.
- D. Rate of settling, dilute-suspension procedure.

Apparatus:

- (2) A 25-cc. pipette, evaporating dishes, balance sensitive to 0.1 g., 100-cc. stoppered glass graduate, putty knife, spatula with blade tapering from $\frac{3}{4}$ to $\frac{1}{2}$ in. at the end, standard ring, watch with second hand, and dry silica brick.
- (3) When tested by the procedure of paragraphs (6 to 8) with the standard silica fire clay, the silica brick shall show a water-retention value of 40 to 50 sec.
- (4) The standard ring for use in the water-retention test shall be made of brass and shall be of the design shown in Fig. 15.

Standard Sample:

- (5) For a primary standard of comparison, silica fire clay shall be used. The primary standard will initially consist of 500 lb. and is to be used sparingly. Secondary standards, for use at the control laboratories, may be prepared by comparison with the primary standard. In using the standards, the following method shall be followed: Weigh out 83.3 g. to the nearest 0.1 g., add 25 cc. of distilled water with a pipette, and mix thoroughly. When the test

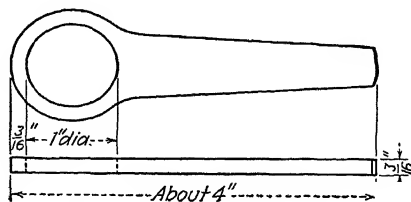


FIG. 15.—Mold for clay testing.

samples have been brought to the same consistency as the standard sample, prepared in this manner, they are of the proper consistency for the spreading and water-retention tests.

A. Water-retention Test.

- (6) The standard sample shall be prepared as described above. One hundred (100) g. of the sample test shall then be weighed out and placed in an evaporating dish. Distilled water shall be added from a glass graduate, working it into the sample until the mixture is of the same consistency as the standard. The amount of water added shall be recorded. If the proper end point is passed, more dry sample (weighed) shall be added, making proper correction on the amount of water reported. Before proceeding, all samples containing either sodium silicate, other chemical bonds, or an organic plasticizer shall be allowed to stand for 30 min. after being tempered. It may be necessary to add more water after the 30-min. period, to bring the mortar back to the proper consistency, in which case the total amount of water shall be reported.
- (7) The standard ring shall be laid flat on a silica brick. Then enough of the test sample to fill the ring shall be placed in it with a spatula, and the excess shall quickly be struck off level with the top of the ring. The ring shall be left in place, and the number of seconds which elapse from the instant the sample was placed in the ring until the "shine" disappears from the top of the pat shall be recorded.
- (8) The "shine" disappears due to the absorption of water from the pat by the silica brick. Most silica fire-clay samples will lose the top film of moisture in 18 to 60 sec.

B. Spreading Time and Working Qualities.

- (9) Enough wet cement prepared as in paragraph 6 shall be picked up with the spatula to fill the standard ring about twice—this will be about 1 in. along the blade and about $\frac{1}{2}$ in. thick. This portion shall be spread as far as it will go on a silica brick, using approximately a $\frac{1}{2}$ in. thick spread, 1 to $1\frac{1}{4}$ in. wide. Then the material shall be smoothed with a backward motion. Then it shall be scraped together into a pile and spread as before on the same

area with one forward and one backward motion. Then it shall again be scraped into a pile and the spreading out and piling up procedure repeated until the mortar becomes too difficult to spread. The number of seconds of spreading time shall be recorded.

- (10) The "feel" or working qualities of the cement shall be observed. Finally the pat shall be scraped from the brick to determine whether it leaves a clean or dirty surface. The most plastic mixes leave a dirty surface.
- (11) Another portion of the prepared mortar shall be spread out on a silica brick and a putty knife placed flatly against it; the blade shall be quickly withdrawn perpendicularly from the brick, to determine whether or not the mortar pulls free from the brick.

C. Rate of Settling: Normal Procedure.

- (12) For silica fire clay, 100 g. of the dry sample placed with 60 cc. of distilled water in a 100 cc. stoppered glass graduate shall be shaken well for 3 min. and allowed to stand. The percentage of clear water after 1 hr. and after 24 hr. shall be reported as percentage settle.
- (13) For ground fire clay, the procedure shall be the same as in paragraph 12, except that 80 g. of dry sample and 64 cc. of water shall be used.
- (14) For furnace chrome, the procedure shall be the same as in section 12 except that 120 g. of the sample shall be used. If there is a formation of a cloudy suspension which is not definitely composed of chrome particles, its volume shall be reported. In this case the per cent settle shall be calculated as follows:

$$\text{Per cent settle} = \frac{\text{cc. clear liquid} + \text{cc. suspension}}{\text{cc. clear liquid} + \text{cc. suspension} + \text{cc. solids}} \times 100$$

- (15) Mortars containing sodium silicate or other binders or plasticizers shall be prepared as in sections 12 to 14, but shall be shaken again after 30 min. standing, and the settling time counted from then.
- (16) When testing unknown types of mortar or mortars which are not classified as fire clay, silica fire clay or furnace chrome, similarity to one of these three types may be used as the basis for deciding whether to follow the procedure of sections 12, 13, or 14.

D. Rate of Settling: Dilute-suspension Procedure.

- (17) Ten grams of the dry sample shall be placed in a 100-cc. stoppered glass graduate and distilled water added until the total volume of the suspension is 100 cc. The mixture shall be shaken thoroughly for 3 min., and timing shall begin immediately, to record the cubic centimeters of clear liquid (per cent settle) after the elapse of 3, 5, 8, 10, and 15 min.
- (18) **NOTE:**—This dilute suspension settling test is found to give greater differentiation between various silica fire clay samples than procedure C. The method is not suitable for samples containing sodium silicate or other deflocculants. It is applicable to some clays.

Silica Refractories.—The raw materials used in the manufacture of silica refractories are quartzite, or so-called ganister rock, and lime. The principal districts in which suitable ganister is found are Huntingdon, Blair, and Mifflin counties in Pennsylvania, the Devil's Lake District in Wisconsin, and in Alabama. Lesser deposits are found in Colorado, Montana, Utah, and California. Some silica pebble rock deposits in southeastern Ohio are also being used. The ganisters of the main producing districts differ little in chemical analysis. The analysis given in Table 6 for

Pennsylvania ganister might be used equally as well for Wisconsin or Alabama. In general, the accessory oxides are alumina, which should not exceed 1 to 1.5 per cent; lime and magnesia, which occur in small amounts and are good bonding agents; and alkalis, which should be kept as low as possible, not over 0.4 per cent. Iron oxide occurs up to about 1 per cent but is not an injurious oxide. The silica content runs about 98 per cent.

The ganister occurs in floes or talus slopes, in which the blocks may vary in size from a few pounds up to several tons, and in the so-called "solid measure" rock from which the bodies of talus originally broke. Not all quartzites are suitable for the manufacture of silica brick. Not only the chemical composition but also the texture is of importance.

Quartzites that break with a sharp fracture are more desirable than those of a sandy nature. The work of J. S. McDowell¹ on this subject, which was published in 1916, is still authoritative.

The lime that is used for bond in silica brick is the high-calcium type and generally conforms closely to the A.S.T.M. Specification for "Quicklime and Hydrated Lime for the Manufacture of Silica Brick," A.S.T.M. Designation C 49. It is usual to use about 2 per cent lime, as

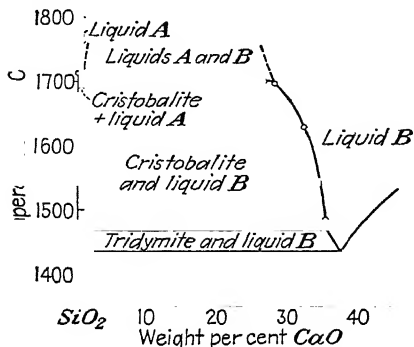


FIG. 16.—Effect of lime on silica refractories.

High-alumina Refractories. Raw Materials.—Within recent years the diasporitic clays of Missouri have assumed a position of increasing importance in the manufacture of refractories. The clays occur in pits or sink holes in limestone formation. Many of these pits contain only so-called flint fireclay, but some grade all the way from the 38.5 per cent alumina typical of the flint clay, through the burley flint of about 44 per cent, burley diaspore of 55 per cent, and No. 1 diaspore of 66 per cent, to occasional patches of material as high as 75 per cent alumina. Diaspore nodules occur in the body of clay, and the variation in alumina content results from the variation in the degree of concentration of these nodules. The purest clay is light gray in color, appearing almost as white as kaolin, but some of it is high in iron oxide and is almost red. The most experienced pit men can sort the clay, on the basis of its alumina content, with uncanny accuracy. Unfortunately, the supply of this type of high-alumina clay appears to be limited. The known pits are being exhausted at a rapid rate under present conditions, and new pits are not being discovered. It would be possible under normal peacetime conditions to use bauxite for the production of high-alumina refractories, but since the aluminum industry now requires all the low-iron bauxite immediately available, it may be that there will be none for refractory purposes. Bauxite was used in refractories a number of years ago, but the refractories produced were decidedly inferior to present day high-alumina refractories. Bauxite is a very high-shrinking material, and it has been difficult to fire the refractories to a sufficiently high temperature to bring them to a state of stable volume. With advance in knowledge, it now seems probable that the present standard of quality can be maintained with bauxite if it can be obtained.

¹ McDOWELL, A Study of Silica Refractories, *Trans. A.I.M.E., Bull.* 119, 1916.

There are also materials other than diaspore used in the manufacture of high-alumina refractories. Kyanite, $\text{Al}_2\text{O}_3\cdot\text{SiO}_2$, both domestic and that from India, fused alumina, and fused mullite are used in special refractories. Topaz has recently been much in the limelight owing to the discovery of a deposit in North Carolina. Estimates of the extent of this deposit differ considerably, even those estimates put out by the government. Topaz, $\text{Al}_2\text{O}_3\cdot\text{SiO}_2\cdot(\text{FOH})_2$, contains much fluorine which is driven off in calcining, carrying some of the silica with it. The calcined product approaches the composition of mullite, $3\text{Al}_2\text{O}_3\cdot 2\text{SiO}_2$, quite closely and may substitute for Indian kyanite to some extent. However, the refractories so far produced have not been so dense as those produced from Indian kyanite and would not be expected to give as good service where resistance to glass or slags is of primary importance. It is unfortunate that no available substitute for Indian kyanite has been found in this country. The domestic kyanite has very different firing characteristics and cannot be substituted for the Indian variety where density of the refractory is of importance.

Tabular alumina, a dense product of nearly pure alumina, and electrically fused alumina are also used in the manufacture of some high-alumina products.

SILICA REFRACTORIES

Table 8, Column (1)

The composition of silica brick has undergone no substantial change for many years. The mixture used is usually 98 per cent quartzite rock and 2 per cent lime. It is true that the ganister is more carefully selected than formerly, or is washed, so that the alumina is held close to 1 per cent or less. While it was formerly a generally accepted rule that the combined iron oxide and alumina should not exceed $2\frac{1}{2}$ to 3 per cent, it is now known that the iron oxide has very little effect on refractoriness of silica brick. Alumina, on the other hand, in a mixture of silica and lime, is a bad actor and will reduce the refractoriness to a considerable extent. The bulk density of silica brick has been increased by application of the principles of grain sizing and in some cases by power pressing the brick instead of molding them by hand or by drop machine.

It has been a practice for a good many years to deplore the necessity for using silica brick in basic openhearth and electric-furnace roofs. This is an attitude of mind that is hardly justified by the facts. Silica brick, although properly classed as an acid refractory, will withstand the addition by absorption of considerable lime and iron oxide from the furnace dust and fumes, without a material reduction in refractoriness. Figures 16 and 17 show, respectively, that lime and iron oxide in considerable amount may be added to silica without making much change in the ultimate melting temperature of the mixture. It is true that the amount of the liquid phase present in the mixture at temperatures below the liquidus line becomes increased with increasing basic oxide content, but the liquid, particularly in the silica-lime mixtures, is highly viscous. The increase in liquid probably causes the ends of silica brick in high-temperature furnaces to soften for a distance of an inch or so (2.54 cm.) from the hot face, but they do not become sufficiently fluid to flow or drip until the temperature is increased nearly to the melting point of pure silica.

Figure 5 shows the reversible thermal expansion curves for certain refractories. The expansion characteristics of silica brick, after the low-temperature danger zone is passed, are ideal. The long range, from about 1200°F. almost to the melting point of silica, in which the volume changes of silica brick are almost negligible, is evident from the chart. Silica brick are not susceptible to thermal spalling at temperatures above 800°F. Where it is possible to heat the silica slowly and steadily through the temperature range in which cristobalite inverts from the low-temperature to the

TABLE 8

	Silica (1)	Siliceous fire clay (2)	Inter- mediate- heat duty (3)	High- heat duty (4)	Super- duty regular (5)	Super- duty special (6)	Kaolin (7)	50% Al ₂ O ₃ (8)	50% Al ₂ O ₃ (9)
Silica.....	96%	77%	60%	53%	52%	52%	53.5%	46%	42%
Alumina.....	1%	21%	35%	42%	43%	43%	43%	50%	52%
P.C.E.....	31-32	29-30	30-31	32%	33-34	33-34	33-34	35	34-35
Approx. °C.....	1690	1645	1665	1723	1752	1752	1752	1785	1772
Approx. °F.....	3074	2993	3029	3134	3186	3186	3186	3245	3186
Density:									
Lb. per cu. ft.....	104	116	132	130	145	148	130	146	136
Kg. per cu. m.....	1666	1858	2114	2082	2323	2371	2082	2339	2178
Porosity.....	25%	30%	18%	18%	14%	12%	21%	16%	23%
Per cent spalling loss:									
High-heat duty schedule.....	5%	7.5%	0	1%		2%	
Superduty schedule.....	2.7%				
Reheat test; per cent linear change, 5 hr. at									
1400°C.....	+0.8%	+0.2%	-0.5%	±0.5%					-0.7%
1500°C.....	-2.0%
1600°C.....					
1650°C.....					
Load test: 25 lb./sq. in. (1.75 kg./cm. ²), 1½ hr., per cent subsidence at									
1300°C.....	3.0%						
1350°C.....	0.5%	5%	3%				
1450°C.....	5%	3%	1%	4%	6%
Temperature of shear:									
°F.....	2960								
°C.....	1627								

TABLE 8.—(Continued)

[illegible]

high-temperature form, about 400 to 525°F., and maintain the temperature above this range, silica brick will give long service. Silica brick are remarkable for their load bearing, especially at high temperatures. In no other refractory of the alumina-silica series can the furnace be operated so close to the final fusion temperature of the refractory.

Table 7 shows laboratory tests on the various classes and types of brick. Only those tests are listed which are considered most important. For example, no spalling test value is shown in column (3) for intermediate-heat-duty fire-clay brick, since such brick are not intended for use under service conditions that require resistance to spalling. Numbers are used throughout the text to refer to the column in Table 8 where the test data are listed.

The mortar most commonly used for laying silica brick consists of a ground mixture of ganister rock, silica brick bats, and a plastic fire clay. For most applications, the refractoriness of such a mixture is sufficient, but for certain operating conditions it is

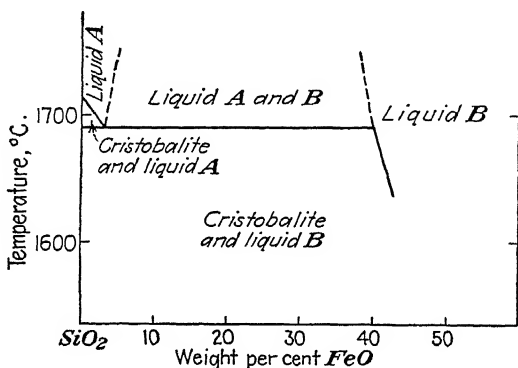


FIG. 17.—Effect of iron on silica refractories.

advantageous to use a mortar whose refractoriness is practically equivalent to that of the silica brick. A mortar has been developed for this service which has the required refractoriness, yet is sufficiently plastic.

SILICEOUS FIRE-CLAY BRICK

Table 8, Column (2)

Fire-clay brick made from siliceous fire clay have two characteristics. The first is the ability to support load at fairly high temperatures, and the second is resistance to spalling. Porosity is generally high, 28 to 32 per cent, and mechanical strength and resistance to abrasion relatively low. Refractoriness as measured by P.C.E. is three to five cones lower than that of other high-heat-duty brick.

HIGH-HEAT-DUTY FIRE-CLAY BRICK

Table 8, Columns (3) and (4)

The development of the power-press process (frequently called dry press) and the vacuum-press process has brought about radical changes in fire-clay brick. While the chemical composition of these brick is practically the same as that of the brick made by hand, the physical properties are decidedly improved. In the old days when

a brick was desired for maximum resistance to spalling conditions, the manufacturer made one of open texture and as light a burn as could safely be transported. Porosities ran as high as 30 to 35 per cent for this type of brick. Today the brick of highest spalling resistance are made by the power-press process and are of high density and hard burn. Porosities run as low as 14 to 16 per cent, and the brick are fired 6 to 10 cones higher than formerly. However, it is not merely the change of process that has brought about the improvement. The brick mix, as fed to the power press, must be properly sized or the desired result will not be attained. This sizing can be accomplished most definitely by grinding the clays, screening, storing the separate fractions in different bins and then recombining the fractions in the proper proportions. The proper sizing may also be attained closely for a given mix by varying the amount of feed to the dry pan, the number and size of open plates in the bottom of the pan through which the crushed clay passes, and the rate of feed and mesh sizes on the screens.

That the advance in resistance to spalling has been a real one is shown conclusively by service records. Some comparative tests recently made in the laboratory on old and on new type brick may be of interest. Three lots were tested. The first lot was made up of the mix and by the process used about 15 years ago and represented about the best possible brick by the old handmade process. The second lot represented brick made by the power-press process without grain sizing. The third lot represented current production of power-pressed brick intended for use under conditions conducive to spalling.

	Process	Spalling loss, %, A.S.T.M. Text C 107-36
Old process.....	Handmade	16.5
3 years ago.....	Power-pressed, unsized	20.0
Current production.....	Power-pressed, sized	3.5

Resistance to slag attack may also be met today by the use of the vacuum-press process brick as well as by brick made by the extrusion process.

There are many kinds of mortar material available for laying fire-clay brick. Fine-ground fire clays, also called dry milled clay, are the old standby. They are made of a mixture of clays and have a P.C.E. of about cone 29 to 32 for High-heat-duty Classification. They act admirably as a cushion between the brick to take up irregularities and give a full bearing surface. Some of these clays have an amount of shrinkage at furnace temperatures that tends to open up cracks through the joints, and they do not bond the brick together except for a narrow band on the inside hot edge of the brick. Because of this looseness of bond, the salvage of brick from an old furnace may be high. Where cracks in the joints are a disadvantage, as in the case of gas leakage, slag penetration, or mechanical weakness, or where arch spalling is troublesome, it is well to use some type of high-temperature bonding mortar. Many kinds are available. Hot-setting mortars are usually mixtures of plastic fire clay and calcined clay. They represent a distinct advance over fine-ground raw fire clay in that they shrink much less and also give a better bond between the brick. As in the case of fire clay, however, the brick are bonded only for a short distance from the hot face. Gas leakage and slag attack are less troublesome with heat-setting mortar than with ordinary fire clay. Air-setting bonding mortars made with fire-clay base and shipped wet or dry are usually rather low in refractoriness but of excellent workability and strong bond. Air-setting bonding mortars made with a high alumina base,

usually shipped in a plastic, ready-mixed condition, have a refractoriness practically equal to high-heat-duty fire-clay brick. They show little if any shrinkage under normal conditions if used in thin joints, they bond the brick tightly from hot to cold face, and they make the joints as nearly gas- and slag-tight as are the brick themselves. Chrome-base mortars, shipped wet or dry, make strongly bonded joints which are highly resistant to attack by slags, especially those high in iron oxide. They are not recommended for use under strongly reducing conditions at high temperatures.

SUPERDUTY FIRE-CLAY BRICK

Table 8, Columns (5) and (6)

The development of the power-press process to its present high standard of usefulness has made possible the development of brick of a new class known as superduty fire-clay brick. The alumina content of high-heat-duty fire-clay brick usually lies below 42 per cent, that of superduty fire-clay brick between about 43 and 45 per cent, while theoretically pure calcined kaolinite is 45.9 per cent alumina. The difference between the alumina content of some high-heat-duty and superduty brick is relatively small and is due to the lower content of accessory oxides of the fire clays used in making superduty brick.

The refractoriness of high-heat-duty brick, as measured by the P.C.E., is normally cone 32 to 32½, that of superduty fire-clay brick cone 33 to 34, a difference not greater than two cones, or about 60°C. (108°F.), but the difference in effective refractoriness is much greater than this small cone difference would indicate. While the relatively slight increment in alumina content and P.C.E. would not normally be expected to have any great effect on service, the difference in physical properties between superduty and high-heat-duty fire-clay brick is substantial. Furnaces have been operating so close to the maximum permissible temperature for the refractories that an increase in effective refractoriness of just a few degrees often results in a material increase in the life of the furnace.

The resistance to spalling conditions shown by superduty brick is one of its outstanding characteristics. The increased resistance to spalling has been definitely established by numerous comparative service records. Measured by the same laboratory spalling test, the superduty brick show zero loss as compared with 2.5 to 10 per cent for the most spall-resistant high-heat-duty brick (A.S.T.M., Test C 107). The test usually used for superduty brick, however, was developed particularly to test brick of this class and is much more severe than that used for high-heat-duty brick. The panels of brick in this superduty test (A.S.T.M., Test C 122) are preheated for 24 hr. at 3000°F., a temperature 90° higher than the former test. The average spalling loss in this more severe test is probably about 3 per cent for all brands of this class, and the maximum allowed by the A.S.T.M. Specification C 27 is 4 per cent.

Another property of primary importance in the superduty brick is volume stability. Service records have proved its exceptional merit in this respect. The A.S.T.M. specification requires that the brick show less than 1 per cent linear shrinkage in the 2912°F. reheat test (A.S.T.M., Test C 113 Schedule C).

Certain of the superduty fire-clay brick are the densest fire-clay brick on the market today (acidproof brick excepted). Porosities range as low as 10 per cent and seldom exceed 16 per cent. One brand showed an average of 13.7 per cent for about 150 tests made in 1938, with all monthly averages between 12.6 and 14.7 per cent. This low porosity gives the brick a decided advantage in resistance to slag attack.

It should be remarked that superduty brick generally present a slightly rough appearance. There is some surface crazing and cracking, and usually the better brick from a service standpoint has a poorer looking surface.

Two types of superduty fire-clay brick [(5) and (6)] are available. The distinction between the two types is largely a matter of degree of crystalline conversion, or stability.

In the operation of three-phase electric steel furnaces, during slack business periods, superduty brick have been found to give more economical operation in the roofs than silica brick. They are also used at times in the roofs of three-phase nickel furnaces, and three-phase arc furnaces operating on ferrochrome.

Another type of brick that lies within the same range of alumina-silica as the superduty fire-clay brick is that made from kaolin (7). Crude kaolin shrinks to a greater extent during firing than do most fire clays. This fact introduces manufacturing difficulties which are difficult and expensive to overcome. One of the most widely known brick of this type is reported to be made largely of kaolin calcined to a very high temperature, with some crude kaolin for bonding material. The brick is strong and dense, has a P.C.E. of about cone 34 and a resistance to spalling in the same range as that of superduty brick made of fire clay. If used to a temperature higher than that to which it was originally fired, about 2900°F., the continued shrinkage characteristics of kaolin again become evident. This shrinkage may or may not be harmful, according to conditions.

HIGH ALUMINA BRICK

Columns (8) to (18)

Brick of the 50 per cent alumina class are customarily made from diasporitic clays in combination with clays of the type used in superduty or high-heat-duty brick. They are used less generally than the superduty brick, probably because of the price differential, but for certain services their higher refractoriness justifies their installation. Brick of the 50 per cent alumina class are made in at least three modifications. The first (8) is similar in character and properties to the superduty brick but has a P.C.E. about one cone higher. It may not be quite so stable in volume as superduty brick, owing mainly to the shrinkage of the diasporite, although this point is open to discussion on the basis of both service records and laboratory tests. The second type (9) is a stronger, tougher brick, of smooth surface and greater apparent density; it is used where slag attack is the primary consideration. This type of brick is not so resistant to spalling nor so stable in volume as the first type, but its greater resistance to the attack of slags justifies its use. The third type (10), made from Pennsylvania nodular clays, has been in use longer than either of the first two. The toughness and strength of this product make it strongly resistant to abrasion. Its service is confined largely to rotary cement kilns.

In the 60 per cent alumina class, two standard types as well as some special types are available; all the 60 per cent alumina brick have about the same P.C.E., cone 36 to 37 (3300°F.). Of the standard types, one (11) is highly resistant to spalling and is in the same class in this respect as superduty brick. In laboratory tests, brick of this type are stable in volume even at extremely high temperatures. In general they show a slight expansion when reheated for 5 hr. at 2912°F. It is not regarded as definitely established that they are as stable in volume under service conditions as the laboratory tests would indicate. The second type (12) is a tougher brick, more resistant to abrasion and slag action but less resistant to spalling conditions than brick of the first type. Both types of brick are made from combinations of Missouri diasporite and fire clay, and their different characteristics result from the manner in which the components are sized and combined.

A special 60 per cent alumina brick is available which is highly resistant to load under soaking heats. This brick (13) is made from fused alumina, diasporite, and fire

clay. Still another type, the so-called sillimanite brick (14) (made from Indian kyanite and domestic fire clay), falls in the 60 per cent alumina class. These bricks are exceedingly stable and highly resistant to chemical attack and solution by soda, lime, and most commercial glasses in the molten condition. They are attacked rather readily by metallic oxides—particularly those of iron and lead.

In the 70 per cent alumina class there are again at least three types of brick available. Two of these are made mainly from diasporic clays. One (15) is of open structure, with about 31 to 34 per cent porosity, and has a marked stability of volume. The second (16) is dense and tough and more resistant to abrasion than brick of the first type. These bricks are close to the mullite composition, which is the only stable form for alumina-silica combinations at high temperatures, as shown in Fig. 13.

The third type of 70 per cent alumina refractory is electrocast mullite. This refractory, being fused and cast, should have its reactions completed and, owing to its high density (porosity nearly zero), it is exceptionally resistant to molten-glass slags. Unfortunately, blocks of this type are not resistant to sudden temperature change and even after the most careful annealing are likely to spall in exposed portions of the furnace structure.

Brick of 80 and 90 per cent alumina (17 and 18) are available for use in furnaces of extraordinarily severe service requirements. They are made from diaspore and corundum with suitable bond and are substantially corundum and mullite when burned.

In general, it should be said that most refractory materials are better if made by the power-press or vacuum-press rather than by the handmade process. This is particularly true of superduty and high alumina brick. It is a decided advantage to use standard shapes from the standpoint of both quality and cost, where this is possible. If special shapes are necessary they should, if possible, be designed so that they may be power pressed.

The mortar materials used for laying high-alumina brick usually present no exceptional problems. The same principles discussed above are merely adapted to the high-alumina mixes.

BASIC REFRACTORIES

Magnesite.—The mineral magnesite, $MgCO_3$, is the source of the most important refractories for metallurgical furnaces that use a basic process. This natural mineral occurs in many parts of the world. Prior to the First World War, practically all the magnesite used in this country was produced in Austria and Hungary. In the United States, deposits in Washington and California have been developed and the former brought to a high rate of production since 1914. Deposits in Russia and Manchukuo have also been developed. Magnesite is now also being produced from sea-water bitterns in California and Japan and from sea water and dolomite in the East and in England. Refractory magnesite is also being produced in Canada by concentration from rock containing calcite, dolomite, and brucite.¹

"Magnesite" dissociates on heating, by the release of carbon dioxide; the product is magnesia (MgO), which develops, either simultaneously or on further heating, a crystalline structure, and is then known as periclase. . . . When dead-burned or sintered, magnesium oxide is resistant to hydration and carbonation at the temperature and pressure of the atmosphere. . . . Strictly the name magnesite should refer only to the mineral $MgCO_3$ but common usage applies it to most of the manufactured products with a classifying prefatory word, *e.g.*, dead-burned magnesite and caustic.

¹ U.S. patent 2208185, M. F. Goudge.

² BRACE, "Magnesite," *Industrial Rocks and Minerals*, Chap. XXIV, pp. 433-447, *A.I.M.M.E.*, 1937.

For the most part these products are actually magnesia. . . . Dead-burned (or sintered) magnesite is the product obtained by calcining magnesite at a temperature of 2640 to 3000°F., which dispels virtually all of the CO₂ and sinters the grains to dense particles suitable for refractory purposes."

There is considerable variation in the composition of magnesites from various deposits. Table 9 gives a few typical analyses:

TABLE 9.—DEAD-BURNED MAGNESITE

Source	MgO	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃	CaO	Igni- tion loss
Chewelah, Washington, regular flotation.....	81.6 88.3	6.9 4.9	1.5 1.5	4.6 2.2	3.3 3.0	
Sea water, California.....	84.5 90.3	5.1 5.1	0.3 0.7	7.3	2.3	
Austria.....	83.0	5.8	1.7		5.0	0.7
Czechoslovakia.....	88.9	0.6	0.5	6.5	2.4	0.2
Russia.....	92.5	1.5	0.5	1.6	2.2	0.9
Manchukuo.....	92.0	3.7	1.0	1.5	1.6	0.6 0.1

Burned Magnesite Brick.—It has been customary to make magnesite brick of a blend of dead-burned magnesites in order to secure the desired properties in the burned product. The magnesite grains are ground to the desired fineness, mixed with about 4 per cent water and a small amount of organic binder, tempered in a wet pan or mixer, and formed into brick on a Boyd or hydraulic press. After drying, they are burned to cone 18 or cone 23.

The burned brick are exceedingly strong and show cold crushing strengths varying from 7000 to over 10,000 lb. per sq. in. Their density is about 170 lb. per cu. ft., and their porosity about 18 to 22 per cent. They are not resistant to spalling and fail by shear in a load test of 25 lb. per sq. in. when a temperature of about 2650 to 2700°F. is reached. Their chief advantage is their density and strongly basic character and consequent resistance to basic slags.

The melting point of pure magnesia is 5072°F. \pm 23°F. (Day-Sosman scale). This is the highest of any of the oxides listed by F. C. Kracek,¹ with the exception of thoria which is estimated at 5520°F.

The mean specific heat of magnesite brick is high compared with most refractories. Values are shown in Table 1.

The thermal conductivity of magnesite brick also is high in comparison with fire clay, silica, and chrome brick values as given in Table 2 and Fig. 4.

Magnesite brick tend to react with other refractories at high temperatures, particularly those of an acid character. Table 10, taken from "Modern Refractory Practice," gives temperatures at which reactions occur between brick of different types. Reactions may occur between brick and mortar at lower temperatures.

Magnesite brick will hydrate in contact with steam. This fact makes it necessary to exercise caution in storing magnesite brick, except on dry floors if they are near furnaces, to avoid using an excess of water in bonding mortar and also to avoid leaky water cooling pipes in furnaces.

¹ KRACEK, Melting and Transformation Temperatures of Mineral and Allied Substances, *Geo. Soc. Am., Spec. Papers* 36, Sec. 11, "Handbook of Physical Constants," edited by F. Birch, pp. 139–174, 1942.

TABLE 10.—REACTION TEMPERATURES BETWEEN DIFFERENT REFRACTORIES
(Degrees centigrade)

Type of refractory brick	Silica		Fire clay		High alumina (70 % Al_2O_3)		Chrome		Magnesite		Forsterite	
	A	B	A	B	A	B	A	B	A	B	A	B
Silica.....	1500	*	1600	*	No reaction		1500	1600	1700	1700
Fire clay.....	1500	*	No reaction		1600	*	1400	1500	1500	1600
High-alumina (70 % Al_2O_3)...	1600	*	No reaction		1600	1600	1500	*	1700	1700
Chrome.....	No reaction		1600	*	1600	1600	No reaction		No reaction	
Magnesite.....	1500	1600	1400	1500	1500	*	No reaction		No reaction	
Forsterite.....	1700	1700	1500	1600	1700	1700	No reaction		No reaction		No reaction	

A Reaction first observed.

B Reaction first became damaging.

* Not damaging.

1400°C. = 2552°F. 1500°C. = 2732°F. 1600°C. = 2912°F. 1700°C. = 3092°F.

Metakase and Chemically Bonded Magnesite Brick.—Within recent years, chemically bonded magnesite brick have been introduced to the trade and have gained increasingly large fields of usefulness. These brick generally contain some chrome ore mixed with magnesite, and this combination produces a radical change in properties. As indicated above, they depend upon a chemical bond for their mechanical strength and are unburned.

Many of the chemically bonded magnesite brick are of the metal-encased type. The brick are covered on three sides with sheet steel. They have the high refractoriness and slag-resisting qualities of burned magnesite brick and are much more resistant to spalling than burned brick. They are used extensively in openhearth furnace walls, electric-furnace side walls, and copper-reverberatory furnace roofs.

Chrome.—Thirty years ago chrome brick were very sparingly used in furnace construction. They were used mainly as intermediate courses between magnesite and silica brick to prevent chemical reaction between them. Today chrome brick are very widely used and with chrome ramming mixtures for furnace bottoms and furnace chrome for patching or fettling material are highly important materials for furnace construction.

Commercial chrome ores vary greatly in composition. They consist primarily of a complex spinel, $\text{RO.R}_2\text{O}_3$, based on the well-known 1:1 ratio, but with four components instead of two. The formula might be given $\left[\begin{smallmatrix} \text{FeO.Cr}_2\text{O}_3 \\ \text{MgO.Al}_2\text{O}_3 \end{smallmatrix} \right]$ in which the FeO and MgO and correspondingly the Cr_2O_3 and Al_2O_3 vary over wide limits yet maintain the 1:1 ratio of the RO component to the R_2O_3 component.

In commercial chrome ores, there is also a fairly large proportion of impurities in the form of magnesium silicates: talc, serpentine, olivine, and others. Typical chemical analyses of the chrome ore used in refractories are given in Table 11. Ores in which the Cr_2O_3 content exceeds 46 per cent are commonly used for metallurgical or chemical purposes and not for the manufacture of refractories. In general, the combined Cr_2O_3 and Al_2O_3 should about equal or exceed 60 per cent in ores suitable for refractory purposes. If this condition is met, ores having a Cr_2O_3 content as low as 30 per cent make quite satisfactory chrome refractories. Satisfactory ore is found in

southern Rhodesia, Greece, New Caledonia, India, Cuba, the Philippines, Russia, and Turkey. There are large bodies of ore in the United States in Montana, California, and Wyoming, but these require concentration before they are usable and up to the present time these ores have not been used for refractories. The grinding necessary for purification puts them in too fine a state of subdivision to be suitable for brickmaking purposes. They should be suitable for furnace chrome if they could be made available at competitive cost.

TABLE 11.—CHROME ORE AS USED IN REFRACTORIES

Source	Cr ₂ O ₃	SiO ₂	Al ₂ O ₃	FeO	MgO	CaO	Ignition loss	Total
Cuba.....	31.8	6.3	27.2	13.0	18.5	0.6	2.2	99.6
Cuba.....	36.9	2.5	28.2	13.5	17.1	0.3	0.8	99.3
Greece.....	40.0	4.6	21.9	15.2	16.7	0.7	1.4	100.5
India.....	46.7	3.8	17.3	18.8	13.1	ND	2.0	100.7
Philippines.....	33.9	4.4	29.5	12.8	17.9	0.2	1.3	100.0
Rhodesia.....	45.4	8.2	13.4	15.8	15.1	ND	3.1	101.0
Russia.....	41.4	7.5	18.2	14.4	16.5	ND	3.8	101.8

Chrome ore has been found and mined from time to time in the eastern part of the United States, but it occurs in small pockets or lenses. No large deposits are known.

There have been great advances in technical knowledge concerning chrome brick in the last decade. Formerly the brick were made by grinding the chrome ore in a wet pan with about 3.5 to 6 per cent water, molding the brick on a Boyd press, and burning them to about cone 16. Brick made in this manner possessed, of course, the chemically neutral characteristics that made them satisfactory as an intermediate course between silica and magnesite brick, but they were highly sensitive to rapid temperature change and failed under a load of 25 lb. per sq. in. at temperatures varying from 2350 to 2600°F. As a result of the increase in technical information, it was found that most of the low-melting silicate impurities in the chrome ore could be converted into forsterite, 2MgO.SiO₂, by adding magnesite to the brick mix and firing. Were the magnesite not added, the silicate impurities in the chrome ore would be converted largely to clino-enstatite, MgO.SiO₂, on firing. Since forsterite has a melting point of 3470°F. and clino-enstatite one of approximately 2840°F., it is evident that chrome brick bonded with clino-enstatite will fail by softening or shearing at a much lower temperature than chrome brick bonded with forsterite. The practice of converting the low-melting silicates to forsterite by the addition of magnesite is used in many chrome brick on the market today. Brick made in this manner do not fail in the load test at 25 lb./sq. in. until temperatures of 2800 to 3000°F. are reached.¹ The same principle is used in making chemically bonded chrome brick. Brick of this type have high cold crushing strength, 5000 to 6000 lb. per sq. in., are very constant in volume even when heated to high temperatures, 2900 to 3000°F., and show much greater resistance to spalling conditions than do regular burned chrome brick.

Ramming mixes made largely of chrome ore but with minor additions to increase plasticity and cold-setting strength are being quite extensively used in heating-furnace and openhearth furnace bottoms. Finely ground chrome ore with or without the addition of plasticizers is extensively used for patching or spooning openhearth furnace walls.

¹ Method of Making Chromite Refractories, G. E. Seil, U.S. patent 2037600; Refractory, Harvey and McDowell, U.S. patent 2053146; Chrome Ore Refractories, Harvey and Birch, U.S. patent 2077796.

Dolomite.—Dolomite is a mineral that in the pure state consists of magnesium and calcium carbonates in equal molecular proportions. Dolomite is widely distributed and occurs abundantly in many parts of the United States. Some of the deposits are quite pure and approach the theoretical composition, but many of them are high in impurities and contain an excess of lime. Dolomite is much cheaper than magnesite and is extensively used both in the raw and in the calcined state for patching or fettling openhearth furnace bottoms. The calcined product, being a mixture of magnesia and lime, is very refractory, and it would be among the most useful of refractories were it not for the fact that lime cannot be dead-burned, at least not in commercial kilns. The lime hydrates upon exposure to the atmosphere. Much research work has been done and many patents have been issued covering means of overcoming this shortcoming. It is now common practice to add iron ore and other fluxes to the dolomite in order to form a stable sintered grain for patching openhearth furnace bottoms.

Burned brick made from dolomite have not been successful in service in this country, although they are now used to some extent in England. It is possible to produce brick that will not hydrate under reasonable treatment, but the stabilizing agents added cause high shrinkage when the brick are heated to high temperatures. Such brick are also sensitive to spalling influences.

Special Refractories.—*Silicon carbide* brick are used extensively in applications in which resistance to abrasion at high temperatures or high thermal conductivity are of primary importance. They have proved of advantage, for example, in the side walls of boiler furnaces at the clinker line of chain-grate stokers. They are also used to form the muffles in certain furnaces in which exposure of the material to furnace gases would be deleterious. Silicon-carbide brick are not resistant to molten iron or iron oxide. They also have a tendency to oxidize or decompose under oxidizing conditions at high temperatures, which may be a disadvantage in some applications. Were it not for their high cost, silicon-carbide brick would doubtless be much more extensively used.

Forsterite refractories made from natural olivine and magnesite¹ were introduced several years ago and are finding a gradually widening market. The natural olivine contains some low-melting silicates, and these are largely converted to forsterite by combining with the magnesite during firing. Forsterite brick are being used in certain parts of the roofs of copper reverberatory furnaces, in copper holding furnaces, in steel furnaces, and in rotary kilns.

Zirconia and Zircon, the oxide and silicate of zirconium, respectively, seem to offer good possibilities on account of their high refractoriness. However, the raw materials on the market are mainly recovered from sand and are in too fine a state of subdivision to produce a dense refractory body by present-day manufacturing methods. They are high in cost. In general, brick made from these materials are not resistant to metallic slags. It is reported, however, that they have given good service in phosphate smelting furnaces and in glass furnace feeder ports. They are also useful in the manufacture of small crucibles and other laboratory ware.

Carbon Refractories.—Clay materials containing carbon in some form have been in use for a long time. Carbon itself must be considered a refractory of high grade which can be used wherever the possibility of continuous oxidation is excluded.

Brick were made many years ago from mixture of crushed coke and clay or crushed coke bonded with tar. Such mixtures are usually hand rammed, dried, and fired in muffles with as complete exclusion of air as possible. The carbonization of the tar cements the coke particles into a compact mass. The coke may also be replaced by

¹ BIRCH and HARVEY, *Forsterite and Other Magnesium Silicates as Refractories*, *Jour. Am. Ceram. Soc.*, Vol. 18, pp. 176-192, 1935.

graphite. Refractories of this type combine high heat-resisting power with excellent thermal conductivity.

Mixtures of clay and graphite are used very largely in the crucibles employed in the melting of metals and in steel ladle stoppers.

Alumina spinel, $\text{MgO} \cdot \text{Al}_2\text{O}_3$, offers much promise as a refractory, but has not made much headway because of high cost and manufacturing difficulties. It has a high melting point. No low-melting eutectics exists between pure alumina and magnesia. During the First World War, brick made from bauxite and magnesite were used quite extensively, but later their use was discontinued and it has not been revived.

Insulating Refractories.—The use of insulating firebrick has increased rapidly within the last few years. There is a great advantage in the low thermal conductivity and low heat capacity of these brick, and where there is no slag and the temperature is not too high, considerable economies can be effected by their use. The A.S.T.M. has classified insulating block and insulating firebrick on the basis of density and of constancy of volume when subjected to a reheat test. The five classes are shown in Table 12.

TABLE 12.—GROUPING OF INSULATING FIREBRICK

Group identification	Permanent linear change not more than 2% when tested at	Bulk density, lb. per cu. ft., not greater than
16	1550°F. (846°C.)	34
20	1950°F. (1065°C.)	40
23	2250°F. (1230°C.)	48
26	2550°F. (1400°C.)	52
28	2750°F. (1510°C.)	60

Brick in group 16 are intended for use at temperatures up to approximately 1600°F. Natural diatomaceous earth and certain light-weight clay refractories meet this classification. Groups 20, 23, and 26, to be used at temperatures of 2000, 2300, and 2600°F., respectively, include brick that are usually made from fire clay mixed with some type of organic material such as sawdust or coke, which is burned out in the firing process, leaving a porous structure.

Group 28 brick are made from high-grade kaolin or high-alumina materials.

Still another type of insulating brick is made from silica ganister with an organic burnout. These bricks are widely used for insulating the silica brick crowns of glass tanks.

TABLE 13.—APPLICATION OF THE COMMON TYPES OF REFRACTORY BRICK AND OF INSULATION IN FURNACE CONSTRUCTION¹

Type of furnace	Application	Kinds of brick used (alone or in combination)
Air furnace.....	Bottom	1-A, 1-B ^a , 2-B
	Side walls	1-A, 1-B
	Roof	1-A, 1-B
	Stack base	1-B
	Stack	1-C, 1-D
Annealing furnace.....	Lining	1-B, 1-C, 6-A
	Combustion chamber	1-A, 1-B, 2-A, 2-B
	Backing	1-C, 1-D, 6-B
Arches, sprung.....	Complete	1, 2, 3, 6-A, 7
Arches, suspended.....	Complete	1-A, 1-B, 2, 3, 4, 5, 6-A, 7
Bake oven.....	Combustion chamber	1-A, 1-B, 2-A, 2-B
	Lining	1-A, 1-B, 1-C
	Backing	1-C, 1-D, 6-A, 6-B
Blast furnace:		
Copper.....	Crucible	1-B, 1-C ¹ , 4, 5
	Shaft	1-B, 1-C
	Settlers	4, 5
Iron.....	Hearth and bosh	1-B
	Inwall	1-B
	Top	1-B
	Piping	1-C
Lead.....	Crucible and shaft	1-B, 1-C
Blast-furnace stove, iron....	Lining	1-B
	Dome	1-A, 1-B
	Well wall	1-A, 1-B, 2-A, 2-B
	Checkers	1-A, 1-B, 2-A, 2-B
Boiler setting.....	Combustion chamber	1-A, 1-B, 2
	Arches and side walls	1-A, 1-B, 2
	Bridge walls	1-A, 1-B, 2
	Baffles	1-B
	Second pass	1-B, 1-C
	Backing	1-C, 1-D
Carbon baking furnace.....	Lining	1-A, 1-B

¹ In some cases the applications given may appear to be conflicting. It must be borne in mind that both furnace designs and operating conditions vary considerably; consequently the refractories requirements vary correspondingly.

^a 1. Fire-clay brick

- 1-A Superduty
- 1-B High-heat duty
- 1-C Intermediate-heat duty
- 1-D Low-heat duty

2. High-alumina brick

- 2-A 50 % alumina class
- 2-B 60 % alumina class
- 2-C 70 % alumina class
- 2-D 80 % alumina class
- 2-E 90 % alumina class

3. Silica brick

- 4. Magnesite brick
- 5. Chrome brick
- 6. Insulation

6-A Insulating refractory brick

6-B Insulating brick

7. Forsterite

TABLE 13.—APPLICATION OF THE COMMON TYPES OF REFRACTORY BRICK AND OF INSULATION IN FURNACE CONSTRUCTION.¹—(Continued)

Type of furnace	Application	Kinds of brick used (alone or in combination)
Carbureter:		
Oil gas.....	Lining	1-B
	Checkers	1-A, 1-B, 1-C
Water gas.....	Lining	1-B
	Checkers	1-A, 1-B, 1-C, 2
Coal gas benches	Retorts	3
	Settings	1-B
Coke oven:		
By-product....	Above floor	3, 6-A, 6-B
	Below floor	1-B
	Regenerators	1-B
	Checkers	1-B, 3
	Flues	1-B, 1-C
	Doors	1-B, 1-C
	Backing	6-B
Beehive.....	Floors and doors	1-B
	Walls and crown	1-B, 3
Converter:		
Copper.....	Lining	4, 5
Nickel.....	Lining	4, 5
Steel.....	Lining	3
Crucible furnace	Lining	1-A, 1-B, 2
Cupola.....	Lining (except melting zone)	1-B, 1-C
	Melting zone	1-A, 1-B, 1-C, 3
	Backing	1-C
Distillation furnace, zinc...	Outer walls	1-A, 1-B
	Center walls	1-A, 1-B, 3
	Roof	1-A, 1-B
Dross furnace (nonferrous).	Bottoms	1-A, 1-B, 4
	Side walls	1-A, 1-B, 2, 4
	Side walls (at slag line)	2, 4
	Roof	1-A, 1-B, 2
Electric furnace:		
Brass.....	Complete lining	1-A, 1-B, 2, 2-B
Steel (acid).....	Bottom	3
	Side walls	3
	Door jambs	1-A, 1-B, 3
	Roof	1-A, 1-B, 2-B, 2-E, 3
Steel (basic).....	Bottoms	4
	Side walls	3, 4
	Door jambs	1-A, 1-B, 3
	Roof	1-A, 1-B, 3, 2-B, 2-E
Enameling furnace.	Complete lining	1-A, 1-B
	Piers	1-A, 1-B, 2-A, 2-E
Enamel frit furnace	Complete lining	1-A, 1-B

TABLE 13.—APPLICATION OF THE COMMON TYPES OF REFRACTORY BRICK AND OF INSULATION IN FURNACE CONSTRUCTION.¹—(Continued)

Type of furnace	Application	Kinds of brick used (alone or in combination)
Forge furnace.....	Lining	1-A, 1-B, 2-A, 2-B
Generator:		
Oil gas.....	Complete lining	1-A, 1-B
Producer gas.....	Complete lining	1-A, 1-B
Water gas.....	Complete lining	1-B
	Wall (at clinker line)	2-A
Glass lehrs.....	Complete lining	1-B, 6-A
	Backing	6-B
Glass pot furnace.....	Lining	1-A, 1-B, 2
	Crowns	1-A, 1-B, 3
Glass tank (continuous and day).....	Regenerators	1-A, 1-B
	Checkers	1-A, 1-B, 2-A, 2-B, 2-E, 4, 7
	Ports	1-A, 1-B, 2, 2-E, 3
	Breast walls	2-E, 3
	Roof	3
	Bottoms and side walls	Flux blocks
Gypsum kettle.....	Complete lining	1-A, 1-B
	Backing	1-C, 6 A, 6-B
Heating furnace.....	Bottoms and several courses in side walls	1-A, 1-B, 4, 5
	Side walls	1-A, 1-B, 2-A, 2-B, 6-A
	Roof	1-A, 1-B, 2-A, 2-B, 6-A
Holding furnace:		
Copper.....	Lining	1-A, 4
Nickel.....	Lining	4
Incinerator:		
Municipal.....	Complete lining	1-A, 1-B
	Backing	1-C, 1-D
Paper mill.....	Lining	1-B
Kilns:		
Cement, rotary.....	Burning zone	2, 4, 7
	Intermediate zone	1-A, 1-B
	Cold zone	1-B, 1-C
	Coolers	1-C
Ceramic, periodic.....	Lining	1-A, 1-B
	Combustion chambers	1-A, 1-B, 2
	Backing	1-C, 1-D, 6-A, 6-B
Ceramic, continuous.....	Lining	1-A, 1-B
	Burning zone (walls and arch)	1-A, 1-B, 2, 3
	Backing	1-C, 1-D, 6-A, 6-B
Dolomite, rotary.....	Burning zone	1-B, 2, 4, 7
	Intermediate zone	1-A, 1-B, 2-C
	Cold zone	1-B, 1-C
	Coolers	1-C, 1-D

TABLE 13.—APPLICATION OF THE COMMON TYPES OF REFRACTORY BRICK AND OF INSULATION IN FURNACE CONSTRUCTION.¹—(Continued)

Type of furnace	Application	Kinds of brick used (alone or in combination)
Kilns: <i>Cont.</i>		
Lime, rotary.....	Burning zone	1-B, 2, 4, 7
	Intermediate zone.....	1-A, 1-B
	Cold zone	1-B, 1-C
	Coolers	1-C, 1-D
Lime, vertical shaft and pot type.....	Lining	1-A, 1-B
	Top lining	1-B, 1-C
	Hot zone and arches	1-A, 1-B, 2, 3
	Piers	1-A, 1-B, 2, 3
Ladles:		
Copper.....	Lining	1-B, 4
Steel, tapping.....	Complete lining	1-B, 1-C, 1-D
Treadwell.....	Complete lining	1-A, 1-B
Litharge furnace.....	Bottom	4
	Lower walls	4
	Upper walls	1-B
	Roof	1-B, 2-B
Locomotive firebox.....	Complete lining	1-B
Malleable-iron furnace.....	Bottom	1-A, 1-B
	Side walls	1-A, 1-B
	Bungs	1-A, 1-B
	Stack base	1-B
	Stack	1-C, 1-D
Melting furnace:		
Aluminum.....	Lining	1-A, 1-B, 2-E
Brass.....	Bottom	1-B
	Lower side wall	4
	Upper side wall	1-A, 1-B
	Roof	1-A, 3
Copper, anode and wire bar.....	Bottom	3, 4
	Lower side wall	4
	Upper side wall	3, 4
	Roof	3
Metal mixer.....	Complete lining	1-A, 1-B, 5
Muffle furnace.....	Complete lining	1-B
	Muffle	3, Silicon Carbide
Openhearth furnace:		
Steel (acid).....	Furnace lining	3
	Checker chamber lining	1-A, 1-B
	Checkers	1-A, 1-B
	Top course checkers	1-A, 1-B, 2
	Backing checker chambers	1-C, 1-D, 6-A, 6-B
	Stack	1-C, 1-D

TABLE 13.—APPLICATION OF THE COMMON TYPES OF REFRACTORY BRICK AND OF INSULATION IN FURNACE CONSTRUCTION.¹—(Continued)

Type of furnace	Application	Kinds of brick used (alone or in combination)
Openhearth furnaces: <i>Cont.</i>		
Steel (basic).....	Bottom	4, 5
	Bottom walls to slag line	4
	Slopes	4, 5
	Upper front walls	3, 4, 5
	Upper back walls	3, 4, 5
	Upper end walls	3, 4, 5
	Corner walls	4, 5
	Ports	4, 5
	Water-cooled ports	2-A, 2-B, 2-C
	Bulkheads	3, 4, 5
	Checker chamber lining	1-A, 1-B
	Checkers	1-A, 1-B
	Top courses checkers	1-A, 1-B, 2-B, 3
	Backing checker chambers	1-C, 6-A, 6-B
	Stack	1-C
Puddle furnace.....	Complete lining	1-A, 1-B
Recuperators.....	Complete lining	1-B
Refining furnace (<i>see</i> Melting furnace)		
Refining furnace (nonferrous).....	Bottom	1-B, 4
	Side walls to slag line	4
	Side walls upper	1-A, 1-B, 2, 3
	Roof	1-A, 1-B, 2, 3
Regenerators.....	Complete lining	1-A, 1-B
	Checkers	1-A, 1-B, 2, 3
	Backing	1-C, 1-D, 6-A, 6-B
Reverberatory furnace:		
Copper.....	Side walls	3, 4
	Roof	3, 4
	Taps	4, 5
Lead.....	Bottom	1-B
	Side walls	1-A, 1-B, 2
	Side walls at slag line	2-C, 4
	Roof	1-A, 1-B, 2
Nickel.....	Lining	In general similar to those used in copper reverberatory furnaces
Roasters.....	Complete lining	1-B
Silicate of soda furnace.....	Bottom	1-B
	Side walls	1-A, 1-B
	Roof	1-A, 1-B, 2, 3

TABLE 13.—APPLICATION OF THE COMMON TYPES OF REFRACTORY BRICK AND OF INSULATION IN FURNACE CONSTRUCTION.¹—(Continued)

Type of furnace	Application	Kinds of brick used (alone or in combination)
Soaking pits (steel).....	Bottom	4, 5
	Lower side walls	4, 5
	Side walls	1-A, 1-B
	Roof	1-A, 1-B
	Backing	1-C, 1-D, 6-A, 6-B
Soda-recovery furnace (paper mill).....	Bottom	5
	Walls	1-B, 5
	Backing	1-C, 1-D, 6-A, 6-B
Softening furnace (<i>see</i> Reverberatory furnace, lead)		
Stacks, chimneys.....	Stack base	1-B, 1-C
	Lining	1-C, 1-D
Still (oil refining).....	Lining	1-A, 1-B, 6-A, 6-B
	Combustion chamber, particularly at slag line	1-A, 1-B, 2
	Bridgewall	1-A, 1-B, 2
	Backing	1-C, 1-D
	Complete lining	1-B
Superheater (water gas).....	Checkers	1-B, 1-C
	Complete lining	1-A, 1-B, 2-A, 2-B
Welding furnace.....	Bottom	1-B, 4, 5
Zinc distillation furnace (<i>see</i> Distillation furnace, zinc)		
Zinc oxide furnace.....	Complete lining	1-A, 1-B, 3

Table 13 is taken from the section on Refractories by McDowell and Guenther in "Handbook of Engineering Fundamentals," edited by O. W. Eshbach, John Wiley & Sons, Inc., 1936. It has been revised and extended by the authors and is used with the permission of the above cited authors, editor, and publisher (1945).

TABLE 14.—AMERICAN REFRACTORIES INSTITUTE STANDARD SIZES OF FIREBRICK
Standard $9 \times 4\frac{1}{2} \times 2\frac{1}{2}$ in. Series

9" straight	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times 2\frac{1}{2}''$
Small 9" brick	$2\frac{1}{2}$ " series	$9'' \times 3\frac{1}{2}'' \times 2\frac{1}{2}''$
9" soap	$2\frac{1}{2}$ " series	$9'' \times 2\frac{1}{4}'' \times 2\frac{1}{2}''$
9" checker	$2\frac{1}{2}$ " series	$9'' \times 2\frac{3}{4}'' \times 2\frac{3}{4}''$
9" split brick	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times 1\frac{1}{4}''$
9" — 2' brick	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times 2''$
9" No. 1 arch	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times (2\frac{1}{2}'' - 2\frac{1}{8}'')$
9" No. 2 arch	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times (2\frac{1}{2}'' - 1\frac{3}{4}'')$
9" No. 3 arch	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times (2\frac{1}{2}'' - 1'')$
9" No. 1 wedge	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times (2\frac{1}{2}'' - 1\frac{7}{8}'')$
9" No. 2 wedge	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times (2\frac{1}{2}'' - 1\frac{1}{2}'')$
9" No. 1 key	$2\frac{1}{2}$ " series	$9'' \times (4\frac{1}{2}'' - 4'') \times 2\frac{1}{2}''$
9" No. 2key	$2\frac{1}{2}$ " series	$9'' \times (4\frac{1}{2}'' - 3\frac{1}{2}'') \times 2\frac{1}{2}''$
9" No. 3 key	$2\frac{1}{2}$ " series	$9'' \times (4\frac{1}{2}'' - 3'') \times 2\frac{1}{2}''$
9" No. 4 key	$2\frac{1}{2}$ " series	$9'' \times (4\frac{1}{2}'' - 2\frac{1}{4}'') \times 2\frac{1}{2}''$
9" feather edge	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times (2\frac{1}{2}'' - \frac{1}{8}'')$
9" neck brick	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times (2\frac{1}{2}'' - \frac{5}{8}'')$
9" end skew	$2\frac{1}{2}$ " series	$(9'' \times 6\frac{3}{4}'') \times 4\frac{1}{2}'' \times 2\frac{1}{2}''$
9" side skew	$2\frac{1}{2}$ " series	$9'' \times (4\frac{1}{2}'' - 2\frac{1}{4}'') \times 2\frac{1}{2}''$
9" edge skew	$2\frac{1}{2}$ " series	$9'' \times (4\frac{1}{2}'' - 1\frac{1}{2}'') \times 2\frac{1}{2}''$
9" jamb brick	$2\frac{1}{2}$ " series	$9'' \times 4\frac{1}{2}'' \times 2\frac{1}{2}''$
Large 9" straight	$2\frac{1}{2}$ " series	$9'' \times 6\frac{3}{4}'' \times 2\frac{1}{2}''$
Large 9" No. 1 Wedge	$2\frac{1}{2}$ " series	$3''$ series $9'' \times 6\frac{3}{4}'' \times 3''$
	3" series	$9'' \times 6\frac{3}{4}'' \times (2\frac{1}{2}'' - 1\frac{7}{8}'')$
Large 9" No. 2 Wedge	$2\frac{1}{2}$ " series	$9'' \times 6\frac{3}{4}'' \times (3'' - 2\frac{3}{4}'')$
	3" series	$9'' \times 6\frac{3}{4}'' \times (3'' - 2\frac{1}{2}'')$
Large 9" No 3 Wedge		$9'' \times 6\frac{3}{4}'' \times (3'' - 2'')$
Flat back straight		$9'' \times 6'' \times 2\frac{1}{2}''$
Flat back sprit		$9'' \times 6'' \times 1\frac{1}{4}''$
No. 1 flat back arch		$9'' \times 6'' \times (3\frac{1}{2}'' - 2\frac{1}{2}'')$
No. 2 flat back arch		$9'' \times 6'' \times (3\frac{1}{2}'' - 2'')$
$9'' \times 6'' \times 2\frac{1}{2}''$ straight		$9'' \times 6'' \times 2\frac{1}{2}''$
$9'' \times 6'' \times 3''$ straight		$9'' \times 6'' \times 3''$
$9'' \times 6'' \times 2\frac{1}{2}''$ No. 1 key		$9'' \times (6'' - 5\frac{3}{8}'') \times 2\frac{1}{2}''$
$9'' \times 6'' \times 3\frac{1}{2}''$ No. 1 key		$9'' \times (6'' - 5\frac{3}{8}'') \times 3''$
$9'' \times 6'' \times 2\frac{1}{2}''$ No. 2 key		$9'' \times (6'' - 4\frac{1}{2}'') \times 2\frac{1}{2}''$
$9'' \times 6'' \times 3''$ No. 2 key		$9'' \times (6'' - 4\frac{1}{2}'') \times 3''$
$12'' \times 6'' \times 3''$ straight		$12'' \times 6'' \times 3''$
$13\frac{1}{2}'' \times 6'' \times 2\frac{1}{2}''$ straight		$13\frac{1}{2}'' \times 6'' \times 2\frac{1}{2}''$
$13\frac{1}{2}'' \times 6'' \times 3''$ straight		$13\frac{1}{2}'' \times 6'' \times 3''$
$12'' \times 6'' \times 3''$ No. 1 wedge		$12'' \times 6'' \times (3'' - 2\frac{3}{4}'')$
$12'' \times 6'' \times 3''$ No. 2 wedge		$12'' \times 6'' \times (3'' - 2\frac{1}{2}'')$
$12'' \times 6'' \times 3''$ No. 3 wedge		$12'' \times 6'' \times (3'' - 2'')$
$13\frac{1}{2}'' \times 6'' \times 3''$ No. 1 wedge		$13\frac{1}{2}'' \times 6'' \times (3'' - 2\frac{3}{4}'')$
$13\frac{1}{2}'' \times 6'' \times 3''$ No. 2 wedge		$13\frac{1}{2}'' \times 6'' \times (3'' - 2\frac{1}{2}'')$
$\quad \times 6'' \times 3''$ No. 3 wedge		$\quad \times 6'' \times (3'' - 2'')$
$\quad \times 6'' \times 2\frac{1}{2}''$ No. 1 key		$\quad \times (6'' - 5'') \times 2\frac{1}{2}''$
$13\frac{1}{2}'' \times 6'' \times 3''$ No. 1 key		$13\frac{1}{2}'' \times (6'' - 5'') \times 3''$
$13\frac{1}{2}'' \times 6'' \times 2\frac{1}{2}''$ No. 2 key		$13\frac{1}{2}'' \times (6'' - 4\frac{3}{8}'') \times 2\frac{1}{2}''$
$13\frac{1}{2}'' \times 6'' \times 3''$ No. 2 key		$13\frac{1}{2}'' \times (6'' - 4\frac{3}{8}'') \times 3''$

TABLE 15.—WEIGHTS OF REFRACTORY AND INSULATING MATERIALS

Material	Average weight, lb. per cu. ft.	Material	Average weight, lb. per cu. ft.
Fire-clay brick:		Dead-burned grain magnesite, in place, when "burned-in" an open hearth or other furnace bottom.....	165-170
Handmade.....	120-129	Finely ground furnace magnesite, in bulk.....	130-140
Extruded.....	125-140	H-W Periklase bonding mortar.....	105-110
Power pressed.....	125-140	Chrome brick:	
Fire-clay materials:		HWR Co.....	180-190
Calced flint clay, in lumps.....	77	H-W Chromex.....	180-190
Finely ground fire clay, in bulk.....	80-85	Chrome materials:	
Plastic clay dobies, dry.....	124	Chrome ore, solid.....	250
High-alumina brick:		Chrome ore, lumps, in bulk.....	140
50 % Al_2O_3	125-135	Finely ground special furnace chrome.....	135-145
60 % Al_2O_3	135-145	Forsterite brick.....	150-160
70 % Al_2O_3	140-145	Forsterite bonding mortar.....	100-105
80 % Al_2O_3	147-152	High-temperature bonding mortars:	
High-alumina material, finely ground high-alumina clay, in bulk.....	90-95	Firebond.....	75-80
Star silica brick.....	100-112	Harwaco Bond.....	120-130
Silica materials:		Thermolith.....	130-140
Finely ground silica fire clay (silica cement) in bulk....	75-80	Duro acidproof brick.....	140-146
Ground ganister (passing a $\frac{3}{8}$ in. screen and including fines).....	80	Insulating brick.....	25-35
Magnesite brick:		Insulating refractory brick....	35-60
HWR Co.....	162-175	Bulk insulation.....	12-30
H-W Periklase.....	162-175		
H-W Magnex.....	180-187		
Magnesite materials:			
Dead-burned grain magnesite, in bulk.....	112		

TABLE 16.—APPROXIMATE FURNACE TEMPERATURES

Kind of furnace	Degrees centigrade	Degrees Fahrenheit
Air furnace (malleable iron):		
Melting chamber, front.....	Up to 1650	Up to 3000
Melting chamber, center.....	Up to 1370	Up to 2500
Melting chamber, at neck.....	Up to 1315	Up to 2400
Base of stack.....	Up to 1315	Up to 2400
By-product coke oven:		
Coking chamber.....	Up to 1200	Up to 2190
Gas chamber.....	Up to 1400	Up to 2550
Copper furnaces:		
Blast furnace.....	Up to 815	Up to 1500
Converter.....	Up to 1200	Up to 2190
Reverberatory smelting furnace, hot zone.....	Up to 1540	Up to 2800
Reverberatory smelting furnace, cool end.....	Up to 1315	Up to 2400
Gas producer:		
Combustion zone.....	1370	2500
Gas leaving producer.....	675	1250
Glass furnaces:		
Plate-glass furnace, between pots.....	1375	2510
Plate glass in pots, working.....	1050	1920
Plate glass in pots, refining.....	1310	2390
Tanks melted for casting.....	1325	2420
Annealing glass.....	440-550	825-1020
Iron and steel furnaces:		
Annealing ovens in foundries.....	925-1000	1700-1830
Blast furnace:		
Front of tuyère.....	1705	3100
Iron at tapping.....	1510	2750
Bessemer converter:		
Steel running into ladles.....	1640	2980
Steel running into molds.....	1580	2875
Openhearth furnace:		
Producer gas entering regenerator.....	590	1100
Producer gas leaving regenerator.....	1200	2190
Air leaving regenerator.....	1100	2010
Waste gases entering air chamber.....	Up to 1430	Up to 2600
Waste gases entering gas chamber.....	Up to 1315	Up to 2400
Waste gases in flue to stack.....	730	1350
Waste gases entering stack.....	650	1200
Refining steel, apparent flame temperature.....	1650-1840	3000-3350
Refining steel, interior wall and roof temperature.....	1530-1620	2785-2950
Steel running into ladles.....	1580	2875
Ingots in soaking pits.....	1200	2190
Lead blast furnace.....	1200	2190
Lead refining furnace.....	1090	2000
Portland cement:		
Clinkering, range.....	1260-1540	2300-2800
Clinkering, average.....	1430	2600

ESTIMATING BRICKWORK: NUMBER OF BRICK

The following figures may be used for estimating purposes. They apply to refractory brick only. A small percentage should be added to net amounts to take care of breakage and cutting.

A square foot of wall requires the following number of 9-in. straight brick:

Thickness of wall, in.	Number of brick required	
	$9 \times 4\frac{1}{2} \times 2\frac{1}{2}$ in.	$9 \times 4\frac{1}{2} \times 3$ in.
$4\frac{1}{2}$	6.4	5.3
9	12.8	10.7
$13\frac{1}{2}$	19.2	16.0

A cubic foot of brickwork requires 17.1 standard 9-in. straight brick of the $2\frac{1}{2}$ -in. series.

A cubic foot of brickwork requires 14.3 standard 9-in. straight brick of the 3-in. series.

1000 standard 9-in. straight brick, of the $2\frac{1}{2}$ -in. series, closely stacked, occupy 58.6 cu. ft.

1000 standard 9-in. straight brick, of the 3-in. series, closely stacked, occupy 70.3 cu. ft.

A GLOSSARY OF TERMS COMMONLY USED IN THE REFRACTORIES INDUSTRY

The preceding tables and the following definitions are from "Modern Refractory Practice" with the permission of the Harbison-Walker Refractories Co.

abrasion. The wearing away of a material at its surface through the cutting action of solids. Refractories are often subjected to the abrasive action of moving charges (such as lumps of lime in a vertical lime kiln), dust-laden gases, or tools used for charging or cleaning.

absorption. As applied to ceramic ware, the ratio of the weight of water which can be absorbed by the ware to the weight of the dry ware. It is expressed as a percentage.

acid refractories. Refractories which contain an appreciable amount of free silica, *i.e.*, silica uncombined chemically with other materials. Silica brick are composed almost entirely of free silica, and are therefore strongly acid; most fire-clay brick contain relatively small quantities of free silica, and are therefore less strongly acid.

amorphous. Lacking crystalline structure or definite molecular arrangement; without definite external form.

arc. As applied to circles, any portion of a circumference.

basic refractories. Refractories which contain either magnesia or lime in considerable quantity. Magnesite brick, which are largely composed of uncombined magnesia, are basic.

batter. A slope of the face of a wall, usually causing a diminution of thickness as the wall ascends; the angle at which a face of a wall slopes from the vertical.

British thermal unit. The amount of heat required to raise the temperature of one pound of water one degree Fahrenheit.

calcination. An industrial heat-treatment to which many rocks or mineral substances are subjected, mainly for the purpose of effecting dissociation, and driving off some constituent in the form of a gas. A familiar example is the calcination of limestone which, in the process, loses weight and is converted into lime, through loss of carbon dioxide. Clays are calcined not only for the purpose of driving off combined water, but also to effect shrinkage.

calorie (small). The amount of heat required to raise the temperature of one gram of water one degree centigrade.

calorie (large). The amount of heat required to raise the temperature of one kilogram of water one degree centigrade.

ceramics. In its narrower sense the term ceramics applies to ware made from clay and "fired" (subjected to the action of heat), and to the art of making clay ware. In its broader and more general application it includes also all refractory materials, cement, lime and plaster, pottery, glass, enamels, glazes, abrasives, and electrical and thermal insulating products made from earthy materials by the application of heat.

chord. As applied to circles, a straight line joining two points on a circumference.

chrome brick. Brick made from refractory chrome ore and usually containing 32 to 44 per cent chromic oxide, Cr_2O_3 .

cold set. The hardening or "setting" of a mortar material which takes place merely upon drying. Bonding mortars containing agents which give them greater strength after drying than is obtainable with a clay or other refractory base alone are said to be cold setting.

colloid. "A condition of the dispersion of a substance in a medium, the peculiar characters of the substance depending upon the size of the particles."—Friend.

Colloidal particles are invisible under the microscope, and their diameters are believed to lie roughly between one-tenth and one-thousandth of a micron. Particles of smaller diameter are of molecular dimensions. Glue, starch, and gelatin are typical colloids, and many clays are at least in part colloidal.

congruent melting. The change of a substance, when heated, from the solid form to a liquid of the same composition. The melting of ice is an example of congruent melting.

corbel. An arrangement of brick in a wall in which each course of brick projects beyond the course immediately below it.

corrosion. Gradual wearing away of a solid body through chemical action, as the rusting of iron. Corrosion of furnace linings through chemical attack by gases, slags, coal ash, or other fluxes occurs in many types of furnace practice.

course. A layer or row of brick as built into a furnace structure.

crystal. A chemically homogeneous solid body having a definite internal molecular structure and if developed under favorable conditions having a characteristic external form, bounded by plane surfaces.

crystalline. Composed of crystals.

dead-burned magnesite. A hard, dense, granular material obtained by calcining magnesite rock at a temperature high enough to form a product inert to atmospheric moisture and carbon dioxide. It is composed essentially of magnesia, MgO , usually with a few per cent of other oxides.

density. The mass of a unit volume of a substance. It is usually expressed either in grams per cubic centimeter, or in pounds per cubic foot.

devitrification. The change in a solid body from a glassy state to a crystalline condition.

dobie. A molded block of ground clay, usually crudely formed.

Dutch oven. A combustion chamber built outside and connected with a furnace.

erosion. Wearing away of a solid through abrasion or corrosion, or through removal of loosened particles by the action of a moving gas, liquid, or solid. In furnace practice, the blowing away by the furnace gases, or washing away by a slag or molten bath, of loosened particles of refractory material from the lining is called erosion. The sand-blast action of dust-laden gases moving at high velocity is an example of erosion caused by abrasion. The washing away of grog particles from a brick by a molten slag, after the bond of the brick has been weakened by chemical attack, illustrates erosion resulting from corrosion.

eutectic temperature. The lowest melting temperature in a series of mixtures of two or more components.

fillet. The concave curved junction of two surfaces which would otherwise meet at an angle. Fillets are used at reentrant angles in the design of brick shapes to lessen the danger of cracking.

firebrick. In its broader application, this term refers to refractory brick of any type; in its narrower and more usual application, it refers specifically to brick made of fire clay.

fire-clay brick. Brick manufactured essentially from refractory fire clays.

flux. A substance or mixture which promotes fusion of a solid material by chemical action. The ashes of many coals, and various other materials, act as fluxes upon refractories.

furnace chrome. Finely ground chrome ore suitable for laying brick or for patching or daubing in furnaces.

furnace magnesite. Finely ground dead-burned magnesite, suitable for use as a joint material in laying magnesite brick, and for patching or daubing furnace masonry.

fusion. A state of fluidity or flowing, in consequence of heat; the softening of a solid body, either through heat alone or through heat and the action of a flux, to such a degree that it will no longer support its own weight, but will slump or flow. Also the union or blending of materials, such as metals, upon melting, with the formation of alloys.

fusion point. The temperature at which fusion takes place. Most refractory materials have no definite fusion point, as they soften gradually over a relatively wide range of temperatures. See also melting point and pyrometric cone equivalent.

grain magnesite. Dead-burned magnesite in granular form, consisting of a mixture of grains usually varying in size from about $\frac{5}{8}$ in. diameter to very minute particles.

grog. Nonplastic material, frequently precalcined, added to a brick or mortar batch to reduce shrinkage in drying and firing.

header and header course. A header is a brick laid with its length perpendicular to the face of a wall. A header course is a course of brick laid in this manner.

high-alumina refractories. In American practice, refractories containing 47.5 per cent or more of alumina. Among the materials used in their production are diaspore, bauxite, gibbsite, kyanite, sillimanite, andalusite, and fused alumina (artificial corundum).

incongruent melting. Dissociation of a compound upon heating, with the formation of another compound and a liquid of different composition from the original compound. For example, mullite, $3\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$ melts incongruently to form corundum, Al_2O_3 , and a silicate liquid.

inversion. A major change in the form and character of a crystalline material accompanying molecular rearrangement. For example, the changes in silica from quartz to cristobalite and from cristobalite to tridymite are inversions.

isomorphous mixtures. Mineral compounds of similar chemical composition and parallel crystal habit are called "isomorphous." Many of these crystallize together in various proportions, forming "isomorphous mixtures."

jamb. An upright structural member forming the side of an opening in a furnace wall.

lime. Calcium oxide, formula CaO ; commercially produced by calcination of limestone.

lintel. A one-piece horizontal structural member spanning an opening in a furnace wall and usually supporting a superstructure.

loss on ignition. A term used in reporting the results of chemical analysis; the loss in weight which results from bringing a sample of a material to a high temperature, after preliminary drying at a temperature just above the boiling point of water. The loss in weight upon drying is called "free moisture"; that which occurs above the boiling point, "loss on ignition." The chemically combined water of clays, and the carbon dioxide of limestones, are usually reported in chemical analyses as "loss on ignition."

magnesite brick. Brick consisting essentially of magnesia, MgO , usually with a few per cent of other oxides.

melting point. In common speech, the temperature at which a solid changes to the liquid condition; technically, the temperature at which crystalline and liquid phases having the same composition coexist in equilibrium. Metals and most pure crystalline materials have sharp melting points, *i.e.*, they change abruptly from solid to liquid at definite temperature. Most refractory materials have no true melting points, as they soften gradually over a relatively wide range of temperatures as an increasing amount of the solid material is changed to the liquid state. See also fusion point and pyrometric cone equivalent.

Metalkase magnesite brick. Magnesite brick with a metal casing which increases their resistance to rapid temperature changes.

mineral. "A mineral species is a natural inorganic substance which is either definite in chemical composition and physical characters or varies in these respects between definite natural limits."—Winchell.

modulus of rupture. Sometimes called transverse or "cross-breaking" strength, is calculated from the formula

$$M = \frac{3WL}{b^3d}$$

M = modulus of rupture in pounds per square inch.

W = total load in pounds at which failure occurs.

l = distance between the supports in inches.

b = width of the specimen in inches.

d = thickness of the specimen in inches.

The load is applied vertically to the top surface of the specimen, midway between two parallel supporting knife edges on which it rests.

monolithic lining. A one-piece lining, or a lining without joints; commonly formed by the use of a material which can be tamped or sintered in place.

neutral refractory. A refractory material which is neither definitely acid nor definitely basic. However, the term is merely relative in most cases, since at high temperatures such a material will usually react chemically with a strong base, functioning as a weak acid; or with a strong acid, functioning as a weak base. Chrome refractories are the most nearly neutral of all the commonly used refractory materials.

nine-inch brick. The standard $9 \times 4\frac{1}{2} \times 2\frac{1}{2}$ -in. straight brick; the unit of measurement of brick quantities in the refractories industry. A single shape, a large

shipment, and the daily or yearly output of a brick plant may all be expressed in readily understandable figures by reducing them to their "9-in. equivalent."

permeability. The property of a porous body which permits a gas or a liquid to flow through it. Permeability is measured by the rate of flow of a standard fluid under definite pressure. The permeability of a body is dependent upon the number, size, and shape of the interconnecting pores. Pores which are sealed have no effect upon permeability.

plastic fire clay. A fire clay which has sufficient natural plasticity to bond together other materials which have little or no plasticity.

plasticity. That property of a material as a result of which it can be molded into desired forms, which are retained after the pressure of molding has been removed. Finely ground clays usually become plastic when mixed with water.

polymorphism. The property by virtue of which some elements and compounds form two or more minerals of different crystal structures and physical properties. For example, carbon crystallizes both as diamond and as graphite. Silica crystallizes as quartz, as cristobalite, and as tridymite—three minerals having distinctive crystal forms and physical properties.

pores. The small voids between the solid particles in a refractory body or brick. Pores are described as "open" if permeable to fluids; "sealed" if impermeable.

porosity. The ratio of the volume of the pores or voids to the volume of the entire brick; usually expressed as a percentage. Generally the volume of pores is obtained by determining the amount of water absorbed by a brick, and therefore does not include the volume of the sealed pores. The result so obtained is an apparent rather than a true porosity.

pyrometric cone. One of a series of pyramidal shaped pieces consisting of mineral mixtures, and used for measuring temperature effect. A standard pyrometric cone is a three-sided pyramid; and, approximately, is either 2 $\frac{5}{8}$ in. high by 5 $\frac{5}{8}$ in. wide at base, or 1 $\frac{1}{2}$ in. high by $\frac{3}{8}$ in. wide at base. Each cone is of a definite mineral composition and has a number stamped upon one face; and when heated under standard conditions, it bends at a definitely known temperature. (See table on page 329.) Pyrometric cones are widely used in controlling the firing treatments for ceramic materials, and in determining their relative resistance to high temperatures.

pyrometric cone equivalent (P.C.E.). A term used as an index of the heat-resisting qualities, or refractoriness, of firebrick and other ceramic materials. Formerly, in speaking of refractories, the terms "fusion point," "melting point," and "softening point" were all used rather loosely for what is now called the "pyrometric cone equivalent."

refractories. Materials of high resistance to fusion suitable for use in furnace construction. While their primary function is resistance to high temperature, they are usually called upon to resist one or more of the following destructive influences: abrasion, pressure, corrosion by slags or other fluxes, or rapid changes in temperature.

rock. A naturally occurring mineral aggregate consisting of one or more minerals. For example, quartzite rock is an aggregate consisting essentially of crystals of the mineral quartz.

rowlock course. A course of brick laid on edge; used in furnace hearths, sills, tops of walls, and sometimes in other furnace parts.

silica brick. Brick made from ganister (quartzite), bonded with lime, and consisting essentially of silica, usually with about 2 per cent of added lime, and containing small quantities of iron oxide, alumina, and alkalis.

silica cement, silica fire clay. Finely ground silica material, used mainly as mortar for laying silica brick.

sill. The horizontal structural member forming the bottom of a door or other opening in a furnace wall.

skewback. The course of brick, having an inclined face, from which an arch is sprung; also, the iron or steel work which supports such a course of brick, and which resists the thrust of the arch.

slag. A substance formed by fusion, in one of several ways:

1. In smelting operations, through the combination of a flux, such as limestone, with the gangue or waste portion of the ore.

2. In refining of metals, by substances (such as lime) added for the purpose of effecting or aiding the refining.

3. By the action of fluxes, such as coal ash, upon refractories.

Slags are generally waste products, although those of some industries form valuable by-products.

softening point. See melting point; also pyrometric cone equivalent.

soldier course. A course of brick set on end; little used in the case of refractories except in the bottoms of some types of furnaces.

solid solution. A homogeneous crystalline phase with a variable composition. The term "solid solution" is often applied to an isomorphous mixture; and sometimes refers to the solution in one substance, of an unrelated substance, by means of an exchange of atoms in the atomic structures of the two substances.

spalling. "Breaking or cracking of refractory brick in service, to such an extent that pieces are separated or fall away, leaving new surfaces of the brick exposed."—A.S.T.M.

specific gravity. The ratio between the weight of a unit volume of a substance and that of a unit volume of some other standard substance, under standard conditions of temperature and pressure. For solids and liquids, the specific gravity is based upon water as the standard; for gases, it is based upon the weight of dry air or hydrogen. Therefore, a solid lighter than water has a specific gravity less than 1; and a solid heavier than water has a specific gravity greater than 1. The total volume of a brick is made up of two parts; first, the solid material, and second, the pore space, sealed and open.

The "true specific gravity" is based on the weight of the solid material, excluding all pores. The "bulk" or "volume specific gravity" is based upon the weight of a specimen as a whole; *i.e.*, the solid material with all its included pores. The "apparent specific gravity" is based upon the weight of the solid material and its sealed pores.

Determinations made by absorption methods yield apparent rather than true values; those made on powders by the pycnometer method yield true values; and those based on over-all measurements and weights give bulk specific gravities. If no sealed pores or voids are present, the apparent and the true specific gravities are the same.

specific heat. The ratio between the amounts of heat required to raise the temperature of a unit weight of a substance and of the same weight of water one degree.

stretcher. A brick laid on flat with its length parallel to the face of the wall. A stretcher course is a course of brick laid in this manner.

thermal conductivity. The property of matter by virtue of which heat energy is transmitted through particles of matter in contact. For engineering purposes, thermal conductivity is measured by the quantity of heat expressed in B.t.u. which will flow in one hour through a square foot of wall one inch thick, for each degree Fahrenheit difference in temperature between the two faces of the wall.

thermal expansion. The increase in volume or in linear dimensions shown by practically all materials when heated. Such expansion is of a temporary character, as the material returns to its original dimensions when cooled. Thermal expansion is not to be confused with volume changes of a permanent character which many materials undergo when heated.

The "coefficient of linear expansion" is the expansion per degree centigrade or Fahrenheit per unit of length.

vitrification. A process of permanent chemical and physical change in a ceramic body, such as a fire clay, at high temperatures; the change is accompanied by the formation within the body, of an appreciable proportion of glass; the most noticeable effect is reduction in porosity. Most unvitified fire-clay refractories have a measurable porosity, and a dull appearance on broken edges. After complete vitrification, through exposure to an extremely high temperature, fire-clay brick are very strong but brittle, the pores have almost entirely closed, and fracture surfaces of brick are usually shiny or glassy in appearance.

SELECTED LIST OF MINERALS AND ROCKS

alumina. Al_2O_3 , the oxide of aluminum. Free alumina forms the mineral corundum; in chemical combinations with SiO_2 and H_2O , alumina forms the various clay and kaolin minerals; in combination with H_2O alone, it forms diaspor, bauxite, and gibbsite.

andalusite. $\text{Al}_2\text{O}_3 \cdot \text{SiO}_2$. Composition: Al_2O_3 62.9 per cent, SiO_2 37.1 per cent. Specific gravity 3.1 to 3.2. Mean specific heat 0 to 100°C ., 0.168. Decomposes on heating, beginning at about 1350°C ., to form mullite, $3\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$ and probably a silicate glass.

asbestos. Commercially the term asbestos refers to any mineral which can be separated into flexible fibers. The most common form of commercial asbestos is chrysotile; empirical formula, $3\text{MgO} \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$. The heat-insulating value of asbestos is high.

bauxite. A high-alumina rock usually consisting of rounded concretionary grains embedded in an amorphous claylike mass, and believed to consist essentially of a mixture, in various proportions, of alumina trihydrate, $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, and alumina hydrate, $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$.

bentonite. A kind of clay characterized by extreme fineness of grain, great plasticity, and great ability to absorb water. It is a hydrous silicate of alumina of variable composition usually containing 5 to 10 per cent of alkalis or alkaline earth oxides.

breunerite. A variety of magnesite containing several per cent FeCO_3 in solid solution. Specific gravity 3.2.

brucite. $\text{MgO} \cdot \text{H}_2\text{O}$. Composition: MgO 69 per cent, H_2O 31 per cent. Specific gravity 2.38 to 2.40. A soft, waxy, translucent mineral; dissociates at high temperatures with the formation of MgO .

calcite. CaCO_3 . Composition: CaO 56 per cent, CO_2 44 per cent. Specific gravity 2.71.

chrome ore. A rock having as its essential constituent the chrome-bearing spinel $(\text{Fe}, \text{Mg})\text{O} \cdot (\text{Cr}, \text{Al}, \text{Fe})_2\text{O}_3$. Chrome ore contains variable amounts of accessory minerals, of which the serpentine minerals are the most abundant. Chrome ore is the raw material from which chrome refractories are made.

clay.¹ "An earthy or stony mineral aggregate consisting essentially of hydrous silicates of alumina, plastic when sufficiently pulverized and wetted, rigid when dry, and vitreous when fired at a sufficiently high temperature."

corundum. Al_2O_3 (Crystalline alumina). Specific gravity 3.95 to 4.10. Melting point 2050°C . (3722°F .). Mean specific heat 20 to 1000°C ., 0.26. Mean coefficient of linear thermal expansion 20 to 1000°C ., 8.5×10^{-6} .

cristobalite. A mineral form of silica; stable from 1470°C . (2678°F .) to the melting point, 1728°C . (3142°F .). Specific gravity 2.32.

¹ Standard Definitions of Terms Relating to Refractories, A.S.T.M. Designation C71-31.

diaspore. $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$. A mineral theoretically containing 85 per cent alumina and 15 per cent water of combination. Decomposes when heated and forms corundum. Specific gravity 3.45.

diaspore clay.¹ "A rock consisting essentially of diaspore bonded by fire clay. Commercial diaspore clay of the purest grade usually contains between 70 and 80 per cent alumina after calcination; the remainder is mainly silica, with small amounts of other oxides."

diatomaceous earth. A hydrous or opalescent form of silica occurring in the form of a soft chalklike very light rock, and consisting mainly of microscopic shells of diatoms and other marine organisms. It is widely used both in the crude and calcined form, for the insulation of furnaces, and for other purposes.

dolomite. $\text{CaO} \cdot \text{MgO} \cdot 2\text{CO}_2$. The mineral dolomite has the approximate composition: CaO , 30.4 per cent, MgO 21.9 per cent, CO_2 47.7 per cent. Specific gravity 2.85 to 2.95. Dissociates in steps from 500 to 900°C. (932 to 1652°F.). The rock called dolomite consists mainly of the mineral of that name and may also contain a large amount of the mineral calcite.

fayalite. $2\text{FeO} \cdot \text{SiO}_2$. Composition: FeO 70.6 per cent, SiO_2 29.4 per cent. Specific gravity 4.0 to 4.1. Melting point 1205°C. (2201°F.).

feldspar. A group of aluminum silicate minerals, of which the most important are: (1) the potash group, of which orthoclase and microcline, $\text{K}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 6\text{SiO}_2$ are the most common, and (2) the soda-lime group, of which albite, $\text{Na}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 6\text{SiO}_2$, and anorthite, $\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$ form the end members of a continuous series of solid solutions. Specific gravity 2.55 to 2.76. Melting points 1122 to 1550°C. (2052 to 2822°F.).

fire clay.¹ "A consolidated sedimentary clay of low flux content usually associated with coal measures." Fire clay is the raw material for fire-clay brick.

flint fire clay. A hard or flintlike fire clay with very low plasticity. It usually breaks with a smooth or shell-like fracture.

forsterite. $2\text{MgO} \cdot \text{SiO}_2$. Composition: MgO 57.3 per cent, SiO_2 42.7 per cent. Melting point 1910°C. (3470°F.). Specific gravity 3.21 to 3.33. No sharp line can be drawn between forsterite and chrysolite. Mean coefficient of linear thermal expansion, 20 to 900°C., 10.5×10^{-6} .

ganister. A dense, high-silica rock (quartzite), suitable for the manufacture of silica brick. Confusion sometimes results from the use of this term, because it is also applied in some parts of the country to crushed firebrick or to mixtures of either crushed firebrick or silica rock, with clay, for use in tamped linings.

gibbsite. $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$. Composition: Al_2O_3 65.4 per cent, H_2O 34.6 per cent. Specific gravity 2.3 to 2.4. Found in some bauxites.

hematite. Fe_2O_3 (red iron ore). Composition: Fe , 70 per cent; O , 30 per cent. Specific gravity 4.9 to 5.3.

kaolin. A white-burning clay usually having kaolinite as its chief constituent. Mica and quartz are the most common accessory minerals. Specific gravity 2.4 to 2.6. The P.C.E. of most commercial kaolins ranges from cone 33 to cone 35, corresponding to 1745 to 1785°C. (3173 to 3245°F.).

kaolinite. $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$. A mineral consisting theoretically of 39.5 per cent alumina, 46.5 per cent silica, and 14.0 per cent combined water. The principal constituent of most commercial kaolins and many fire clays. Specific gravity 2.59.

kyanite. $\text{Al}_2\text{O}_3 \cdot \text{SiO}_2$ (Cyanite). Composition: Al_2O_3 62.9 per cent, SiO_2 37.1 per cent. Specific gravity 3.56 to 3.67. Decomposition begins at about 1325°C. (about 2415°F.) with the formation of mullite and probably a silicate glass.

¹ Standard Definitions of Terms Relating to Refractories, A.S.T.M. Designation C 71-31.

limestone. A sedimentary rock composed essentially of calcite, CaCO_3 or calcite mixed with more or less dolomite, $\text{CaCO}_3 \cdot \text{MgCO}_3$. Specific gravity 2.6 to 2.8.

limonite. $\text{Fe}_2\text{O}_3 \cdot n\text{H}_2\text{O}$ (brown iron ore). Approximately Fe_2O_3 85 per cent, H_2O 15 per cent. Specific gravity 3.6 to 4.0.

magnesioferrite. One of the spinel group of minerals. $\text{MgO} \cdot \text{Fe}_2\text{O}_3$. Composition: MgO 20.2 per cent, Fe_2O_3 79.8 per cent. Rarely found in nature; usually constitutes the red coloring material in magnesite brick. Specific gravity 4.57 to 4.65.

magnesite. A mineral consisting of magnesium carbonate MgCO_3 , and theoretically containing 47.8 per cent magnesia and 52.2 per cent carbon dioxide; also a rock containing the mineral magnesite as its essential constituent. Commercial refractory magnesite rock contains small percentages of other oxides such as lime, silica, alumina, and iron oxide. Specific gravity 2.9 to 3.1.

magnetite. Fe_3O_4 (magnetic iron ore). Composition: iron 72.4 per cent, oxygen 27.6 per cent. Specific gravity 5.17 to 5.18. Strongly magnetic.

mica. A group of rock minerals having nearly perfect cleavage in one direction and consisting of thin elastic plates. The most common varieties are muscovite and biotite.

mullite. $3\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$. Composition: Al_2O_3 71.8 per cent, SiO_2 28.2 per cent. The only compound of SiO_2 and Al_2O_3 stable at high temperatures. Specific gravity 3.15. Mean specific heat 20 to 800°C ., 0.175. Mean coefficient of linear thermal expansion 20 to 1400°C ., 5.4×10^{-6} . Dissociates at 1830°C . (3326°F .) to form alpha-alumina (corundum) and a silicate glass.

nephelite. $\text{Na}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$. A mineral common in igneous rocks. A common reaction product in furnaces wherein slags or vapors of high soda content come into contact with fireclay or high-alumina brick. Stable to 1248°C . (2278°F .) at which temperature it inverts to carnegieite. Specific gravity 2.67.

nodular fire clay.¹ "A rock containing aluminous or ferruginous nodules, or both, bonded by fire clay. In some districts such clays are called 'burley' or 'burley flint' clay."

olivine. A series of minerals having the general formula $2\text{RO} \cdot \text{SiO}_4$, in which RO is MgO , CaO , FeO , and sometimes MnO . More specifically, chrysolite, a continuous mineral series consisting of forsterite, $2\text{MgO} \cdot \text{SiO}_2$ and fayalite, $2\text{FeO} \cdot \text{SiO}_2$ in varying proportions. Specific gravity 3.27 to 3.37, increasing with the amount of iron present.

periclase. MgO (crystalline magnesia). Specific gravity 3.65. Melting point 2800°C . (5072°F .). Mean specific heat 20 to 1000°C ., 0.285. Mean coefficient of linear thermal expansion 20 to 1000°C ., 13.5×10^{-6} . The form of MgO which occurs in magnesite brick.

picotite. A member of the isomorphous series of minerals of the spinel group; intermediate in composition between $\text{MgO} \cdot \text{Al}_2\text{O}_3$ and $\text{FeO} \cdot \text{Cr}_2\text{O}_3$. Specific gravity 4.08.

plastic fire clay.¹ "A fire clay of sufficient natural plasticity to bond nonplastic materials."

pyrite. FeS_2 (iron disulfide). Composition: Fe , 46.6 per cent, S 53.4 per cent. Specific gravity 4.9 to 5.2. Color, brass yellow. The most common sulphide mineral.

quartz. A common mineral, consisting of silica, SiO_2 . Sandstones and quartzites are composed largely of quartz, which occurs also in numerous other rocks. Specific gravity 2.65. Mean specific heat 20 to 1000°C ., 0.263.

quartzite. A rock consisting essentially of quartz grains united by a cement of secondary quartz. In general, quartzites are metamorphosed sandstones. When

¹ Standard Definitions of Terms Relating to Refractories, A.S.T.M. Designation C71-31.

a quartzite rock is broken, the fracture passes through both grains and bonding material; while in sandstone, the fracture occurs entirely in the bonding material. Quartzite is the raw material from which silica refractories are made.

rutile. TiO_2 . (Titanium.) Specific gravity 4.18–4.25. Occurs in most fire clays in small amounts.

serpentine. A group of hydrous magnesium silicate minerals. Chrysotile and antigorite, $3\text{MgO} \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$, are the most common. Specific gravity 2.5 to 2.7. Also, a common rock consisting essentially of serpentine minerals.

siderite. FeCO_3 (ferrous carbonate). Composition: FeO 62.0 per cent, CO_2 38.0 per cent.

silica. SiO_2 , the oxide of silicon. Practically all rocks contain silica in some proportion, either free or combined. Quartz and chalcedony are common silica materials; quartzite, sandstone, and sand are composed largely of free silica in the form of quartz.

sillimanite. $\text{Al}_2\text{O}_3 \cdot \text{SiO}_2$. Composition: Al_2O_3 62.9 per cent, SiO_2 37.1 per cent. Specific gravity 3.24. At about 1530°C . (2786°F .) it begins to decompose into mullite and a silicate glass.

soapstone. A soft magnesium silicate rock consisting mainly of talc.

spinel. A group of minerals of the formula $\text{RO} \cdot \text{R}_2\text{O}_3$, in which RO may be MgO, FeO, MnO, or ZnO or mixtures of them; R_2O_3 represents Fe_2O_3 , Al_2O_3 , Cr_2O_3 , or their mixtures. Pure magnesian spinel, $\text{MgO} \cdot \text{Al}_2\text{O}_3$ contains 28.2 per cent MgO, 71.8 per cent Al_2O_3 . Specific gravity 3.6. Melting point 2135°C . (3875°F .).

talca. $3\text{MgO} \cdot 4\text{SiO}_2 \cdot \text{H}_2\text{O}$. A hydrous magnesium silicate. Composition: MgO 31.9 per cent, SiO_2 63.4 per cent, H_2O 4.7 per cent. Loses half its H_2O at dull red heat and the rest rapidly between 875 to 960°C . Specific gravity 2.6 to 2.8.

tridymite. A mineral form of silica; stable from 870 to 1470°C . (1598 to 2678°F .). Specific gravity 2.26.

vermiculite. A group of micaceous minerals, all hydrated silicates, varying widely in composition. They are alteration products of the micas. When heated, vermiculite exfoliates and increases greatly in volume.

wollastonite. $\text{CaO} \cdot \text{SiO}_2$. Composition: CaO 48.3 per cent, SiO_2 51.7 per cent. Specific gravity 2.9. Inverts at 1200°C . (2192°F .) to pseudo-wollastonite ($\text{CaO} \cdot \text{SiO}_2$). Melting point 1544°C . (2811°F .).

zircon. ZrSiO_4 . Composition: ZrO_2 67.1 per cent, SiO_2 32.9 per cent. Specific gravity 4.7. Melting point 2550°C . (4622°F .). Mean coefficient of linear thermal expansion 20 to 1000°C ., 4.4×10^{-6} .

zirconia. ZrO_2 . Specific gravity 5.8. Melting point 2700°C . (4892°F .). Mean specific heat 20 to 1000°C ., 0.16. Mean coefficient of linear thermal expansion 20 to 1000°C ., 6.6×10^{-6} .

CHAPTER XII

SEPARATION OF SOLIDS AND LIQUIDS FROM GASES

By P. E. LANDOLT¹ AND F. H. VIETS²

The cleaning of air or other gases by the removal of solid or liquid particles is an engineering problem requiring an intimate knowledge of gas engineering, together with a thorough understanding of related plant operations. It also requires a knowledge of the properties of gases and of the corrosive effects of temperature and various substances contained in the gases on the materials of construction of the gas-cleaning apparatus.

Gas-cleaning falls into two general classes: (1) *Dry Cleaning*, for the removal of suspended solids from gases; (2) *Wet Cleaning*, for the removal of either solid or liquid particles from gases. There are many types of apparatus, some of which are adaptable to either class.

The degree or completeness of cleaning required largely determines the type of equipment selected. To some extent, related plant operations become a factor in the selection of such equipment. Finally, questions of first cost and operating expenses must be considered.

The following chapter has been prepared, with due consideration of the problems of the plant engineer, for the purpose of acquainting him with the various types of equipment available, without attempting to draw a conclusion as to the particular equipment to be selected for a given application.

The various types may be classified as follows:

1. Settling chambers or pure-gravity-type separators.
2. Direction changers or differential-inertia-type separators.
3. Impingers or impact-type separators.
4. Dry filters or filtration-type separators.
5. Wet washers or scrubber-type separators.
6. Electrical precipitators or particle-ionization-type separators.

Settling chambers or pure-gravity-type separators are the simplest type of apparatus and depend on a low rate of gas flow. Gases moving at an appreciable velocity carry finely divided particles, which will settle out if the gas velocity is reduced by passing the gases through a greatly enlarged flue section. The particles thus settled must be relatively dense, *i.e.*, having the maximum mass for the minimum surface.

In gravity settling, the fundamental factor is the settling velocity which can be expressed by Stokes' law as

$$v = \frac{gpd^2}{kn}$$

where v is the settling velocity, p the density of the particles, d the diameter of the particle, n the viscosity of the gas, and k a constant. From this expression, the settling velocities for different sized particles are shown to vary from 1.5 ft. per sec. for a 100-micron particle to 0.0003 ft. per sec. for a 1-micron particle.

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A similar expression for centrifugal separators is

$$v = \frac{r}{kn}$$

where w is the angular velocity of the gas, r the radius of the path of the particle, with the other symbols the same as in the previous paragraph. These formulas explain why fume particles of less than 1 micron will not settle out.

The Howard dust chamber is one example of a modified settling chamber and depends for its advantage over a straight settling chamber on the shelves which produce increased settling areas and, of more direct importance, reduce the height of fall of the particles. The apparatus shown in Fig. 1 has a capacity of 4000 cu. ft. per

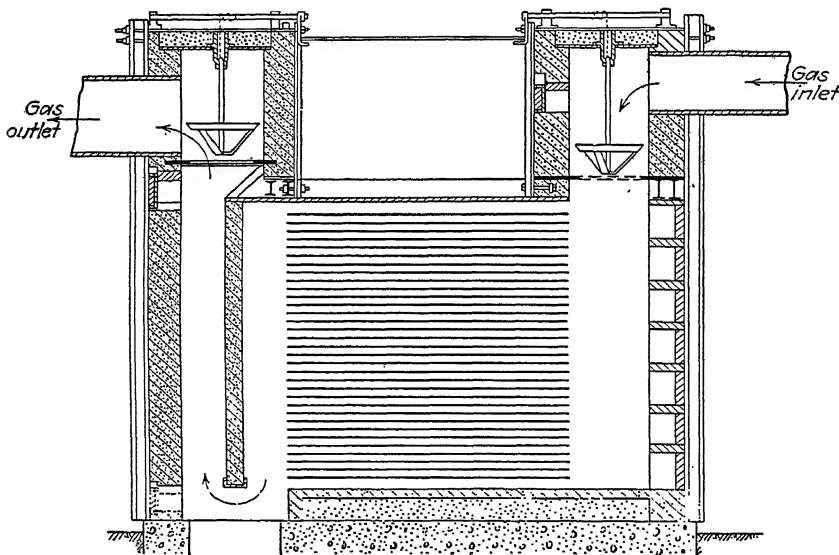


FIG. 1.—Howard dust catcher.

min. at 1000°F. This apparatus has been developed for the cleaning of SO_2 gases from pyrites-fines burners. Its efficiency varies from 60 to 95 per cent. It is usually shut down and cleaned out once a month.

The wire-hung dust chamber is another modification of the settling chamber. In this type, a great number of coarse wires are hung in the chamber and serve to produce obstructions for the particles to impinge upon and to produce small eddies which may promote the settling action.

The advantages of the gravity-type separators may be listed as low cost, both as to first cost and operating cost; low draft loss; and applicable to all reasonable temperatures.

Its disadvantages are large space requirements and limitation in its application and efficiencies.

Differential-inertia-type Separators.—Centrifugal separators have come into considerable prominence in nonferrous metallurgy in recent years and are the most widely

applied form of this type. These may vary from the large-diameter single unit, widely used in the separation of sawdust, etc., and sometimes used on agglomerated or flocculated metallurgical fume, to the small-diameter multiple-tube units used in the separation of finer dusts. All these depend for their separation on the formation of a high-velocity whirl or vortex in a cylindrical tube, from which the gas is drawn off from the center of the vortex. The suspended particles are driven toward the inner surface of the tube by centrifugal force and thence to a suitable exit into a quiescent chamber and conveyed away.

The initial whirl may be given the gas by means of a tangential entrance into the tube or by a system of properly arranged vanes.

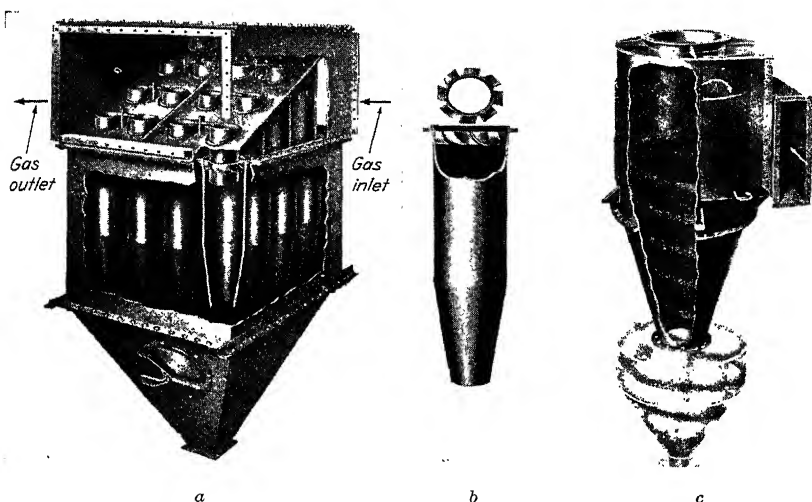


FIG. 2.—Differential-inertia separators: *a*, VG multiclone assembly; *b*, VG collecting tube (Western Precipitation Co.); *c*, Buell Engineering Co. unit.

In these units, power in the form of draft is consumed in moving the gas through the unit, and the efficiency of separation depends on many design features and is a function of the amount of power consumed. The amount of power required to move a given volume through a unit designed for a given efficiency may be expressed thus

where h is the pressure drop across the unit, expressed in inches of VWC, v the gas volume in cubic feet per minute, t the absolute temperature in F° , and k a constant, depending on the design of the unit. In practice h may be 1 to 6 in. of water.

Efficiency may vary from 60 to 98 per cent.

It should be pointed out that this type of separator is limited as to the size of particle that may be collected. Within limits, the smaller the diameter of the tube, the smaller the particle that may be separated by it.

The following table will serve to illustrate this ratio:

COLLECTION EFFICIENCY OF CYCLONES¹

Approximate particle-size distribution of suspended dust (density = 3 g. per cc.)

$\frac{0-5\mu}{26\%}$	$\frac{5-10\mu}{10\%}$	$\frac{10-20\mu}{21\%}$	$\frac{+20\mu}{43\%}$
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Pressure drop—4 in. water. Inlet dust concentrate—2 to 5 grains per cubic foot

Diameter of Cyclone, In.	Per Cent Collected			
6	Total 90	—5 μ	66 + 5 μ	98
9	Total 83	—10 μ	60 + 10 μ	99
24	Total 70	—20 μ	47 + 20 μ	98

The main application is on dry materials.

Some recent work indicates that two cyclones operating in series show considerable improvement in over-all efficiency for a total pressure drop over a single-stage unit having the same pressure drop.

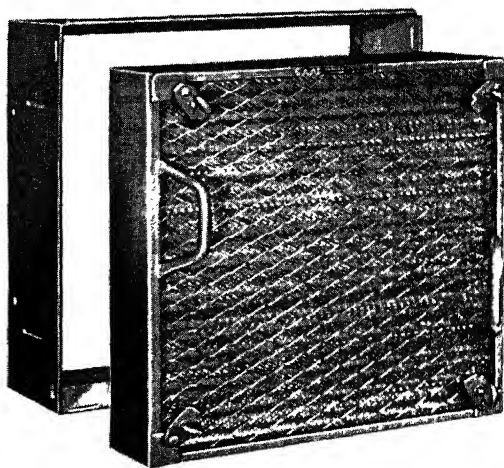


FIG. 3.—Impact-type separator.

Impact-type Separators.—Dry impact separators may be either the small member type such as the wires or chains that are sometimes hung in settling chambers, or they may be the high-velocity types where the inertia of the coarser particles throws them against a surface placed in their path, thus bringing them to rest and allowing them to drop out of the stream before they can be entrained again. An example of this type of equipment is shown in Fig. 3.

¹ "Chemical Engineers' Handbook," chapter on Dusts and Mists, by Evald Anderson, p. 1860, 2d ed., 1941.

Dusts and Mists, by Evald Anderson,

Baffle separators may be considered under this heading. These use baffles to divert the gas by a system of plates or rods that allow the particles to be carried on, because of their inertia, into quiescent zones where they are collected. These also depend on the suspended particles being relatively dense.

The apparatus consists of a steel box containing a series of metallic screens and baffles arranged so that the gas passes through a nozzle-like compartment, causing the gas and dust to attain a high velocity; then this velocity is reduced in a large area following. The dust passes through the screens and drops into a dead-gas space and thence to the hopper for final removal.

This method is based on a difference in inertia between dust and gas, and functions best with large dust particles. It is limited to dust and is not effective for fume.

The pressure drop across the machine varies with the design, but resistance to gas flow up to $1\frac{1}{2}$ in. of water may be expected in commercial equipment of this character. Periodical shaking of the collecting elements may be necessary to remove collected dust and prevent clogging.

Wet-surface impact separators are those which place a wet or sticky surface in the path of the gas and depend on the inertia of the particle to throw it against the treated surface, where it is held. These units may utilize rigid coated surfaces, such as oiled mesh, or containers packed with irregularly shaped objects, oil-soaked. They embody the so-called viscous type of filter used in air cleaning. Their use in the metallurgical and industrial field is limited by their inability to handle heavy dust loadings.

An example is the P. & A. tar extractor which depends on the passage of the gases through a series of perforated baffles. This apparatus is inexpensive, removes 75 to 95 per cent of the tar, requires little attention, and is compact. Back pressure in this apparatus varies from $\frac{1}{2}$ to 1 in. of water.

Another example is a device that consists of "units" of steel frames with perforated plates or coarse mesh screen holding small steel or cast-iron pieces, coated with grease. This apparatus is principally used for air cleaning, for which purpose it is simple and effective. The grease is periodically melted off the iron pieces, the dust separated, and the grease recovered for future use.

These may also consist of liquid surfaces where the gas is blown down against the surface of a reservoir. An example of this is the *Murray washer* developed to remove cinders from stack gases. It depends for its operation on the impingement of a high-velocity jet on a wet surface, usually a bath of water. The gases are turned through 180 deg. in their passage over the water surface. Particles of considerable size and density are removed almost completely.

Its action has been considerably improved by the addition of sprays at the point where the gas is driven down and by a system of spray eliminators at the outlet.

Filtration-type Separators.—This type of apparatus utilizes the principle of filtration wherein the suspended matter is strained out by a close-mesh obstruction. Such apparatus may take various forms, the commonest and most widely used being the cloth filter.

In cloth filters, usually bags, the filter or strainer is flexible. The suspended matter is collected on the fibers of the bag material and is finally dislodged by jarring the bag. The mesh of such a bag must be sufficiently close to prevent the solid matter from passing through. The suspended matter deposited on the cloth fibers assists in the filtration.

Finely divided or fibrous materials such as mineral wool, coke, quartz, asbestos, glass wool, and steel wool are also used for filtration of suspended matter from gases.

Fine-mesh metallic screens are sometimes used, but they are more difficult to clean than cloth screen, and possess no appreciable advantage over cloth under the conditions of operation possible.

A gas cleaner of this type is limited by the following conditions:

1. The temperature at which the bags will lose their strength. For cotton cloth this is at or near 180°F., and for wool somewhat higher. Some glass fabrics have been developed that may come into general use to bring this temperature limit up to approximately 700°F.

2. The chemical nature of the gas. An excess alkaline or acid condition of the gas destroys and impairs the usefulness of the filter cloth.

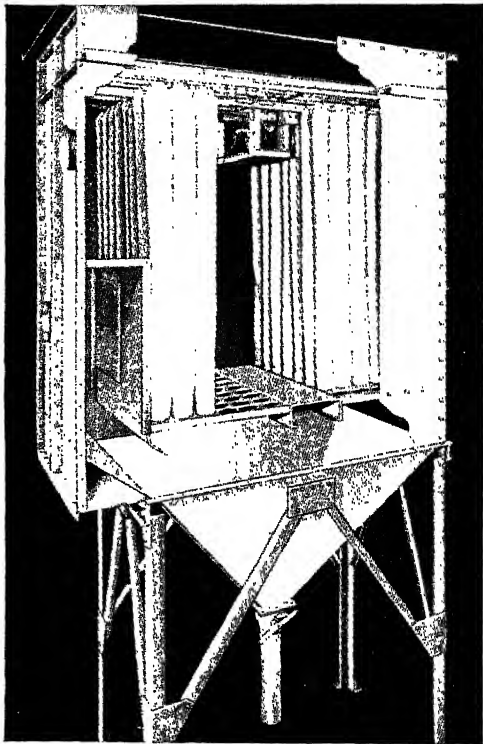


FIG. 4.—Bag filter. (American Foundry Equipment Co.)

3. Moisture in the gas has a tendency to clog the bags and naturally interferes with their proper cleaning. Acid or alkaline conditions are destructive in a moist gas.

4. The pressure drop across such a machine is approximately $\frac{3}{4}$ to 3 in. of water.

An efficiency up to 99 per cent may be obtained. The chief item of maintenance is replacement of the filter bags.

An example of this type of equipment is the *automatic bag filter*. In this type of cleaner, the dust is collected by passing the air or fume through bags of special weave, which are automatically cleaned at definite intervals. The dust is discharged into a hopper at the bottom of the machine.

The collector consists of a number of annular compartments, each containing 12 bags for the collection of the dust. The dust-laden air enters the machine at the bottom of the compartment, making a sharp turn in entering the lower ends of the bags, and thus tending to cause the larger particles to drop into the hopper. The upper ends of the bags are closed, the air or gas passes through the interstices of the cloth, and the dust is collected on the cloth fibers. The cleaned gas passes through to the flue at the top of the compartment.

The bags are cleaned by a violent shaking, the dust dropping into the dust hopper. While the bags are shaken, a current of gas is passed through the bags in the opposite direction of the flow while cleaning the gas. This countercurrent flow aids the bag cleaning.

Another example of this type, the *Sly dust arrester*, is representative of the commercial application of the flat cloth screen as a dust collector. The arrester is usually a rectangular metal box having a series of plane cloth screens vertically placed in the chamber. The screens are continuous and act as one long screen, and are arranged in zigzag fashion to give a larger filtering area.

The dirty gas enters a chamber called the "dust chamber." The gases entering this chamber lose considerable velocity and, therefore, some of the dust. On the clean-gas side is a chamber known as the "fresh-air chamber," which receives the gas from the screens. The fan is usually placed near the fresh-air chamber. The dust is caught in a hopper at the bottom of the arrester. A shaking system is provided which can be operated mechanically or manually to clean the screens when they become clogged.

The small "stocking" dust collector, consisting of a number of bags rotated and automatically cleaned, has found some small applications in the recovery of the minor metals, though its main use is in manufacturing operations.

Scrubber-type Separators.—In such apparatus all the principles applied to dry cleaning and described above may apply, but, in addition, water is used to create a tremendous surface for catching particles, which surface is continuously being removed from the path of the gases.

Where ordinary shower sprays are used in large open chambers or where gas is bubbled through liquid, a proper and sufficient "wetting" of the particles is not effected.

Other forms of apparatus have been devised wherein the gases may be passed up or down through a tower or chamber in which there is some obstruction to flow, such as perforated plates, checkerwork, or rings, on which there is a continuous moving film of water or other liquid. Sprays or whirling nozzles may be substituted for baffles, checkerwork, etc. Examples of such apparatus are the *Brassert and Steinbart tower washers* in iron blast-furnace practice.

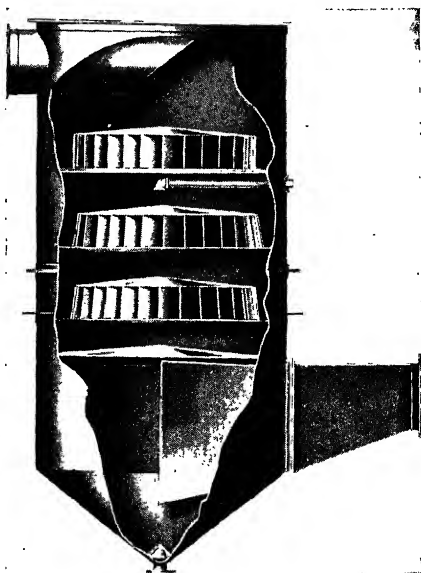


FIG. 5.—Schneible multi-wash tower.

The process of washing or screening or filtering and washing may be further aided by causing either the gas or the sprays or the baffles to be set in motion by mechanical means in such a way as to promote centrifugal action, more perfect distribution of spray, or intimate mixing of the water particles and the gas particles. *Theissen and Feld type apparatus* illustrate this type of equipment.

Good results have also been obtained by conducting the gases at high velocity to an apparatus in which there is a relatively large surface of water and causing the gases to be turned 180 deg. as they impinge on the surface of the water. This type is discussed in the previous paragraph on the *Murray washer*.

Of the *gas washers*, the *Bartlett-Hayward Co. Feld-type scrubber* is probably best known in nonferrous metallurgy. The Feld scrubber operates on the principle that dust particles are forced into contact with a very large surface of water by agitation and are "screened out." The dust particles must pass through the series of "screens" of water. The water particles pick up dust particles and are deposited, largely by centrifugal action.

The scrubber consists of a number of separate washing chambers mounted one on top of the other and rotating on the same vertical shaft. Each washing chamber is in the form of the frustum of a cone, there being five solid shells. The sixth or outside shell is perforated.

The gas enters at the bottom of the shell and the liquid at the top, the gas leaving the scrubber at the top. When the vertical shaft revolves, the cones pick up the water, which is thrown horizontally with considerable velocity through the outer perforated cone. The liquid is projected through the gas space in the form of a fine high-velocity spray through which the gas has to pass.

In this scrubber the minute particles of liquid and gas come in intimate contact, assuming that the gas will not have an opportunity of channeling. Also, other liquids than water can be used as the liquid medium. However, when chemical reactions take place in such a scrubber it becomes a "reaction chamber," and not merely a gas cleaner.

One disadvantage of this scrubber is that if the dust is a by-product the plant operations will have to be extended to handle the liquor containing such dust.

Electrical Precipitators.—The principles that underlie the operation of the electrical precipitator are so different from those governing other previously described types of equipment that this group falls into a unique position by itself. It is applicable not only to all the problems considered thus far, but to any problem where solid or liquid particles are suspended in a gaseous medium.

The degree of cleaning, or percentage removal of solids from the gas, not only can be foretold accurately but any desired percentage efficiency may be designed for, even approaching (but never quite reaching) 100 per cent, or total cleaning. The efficiency varies with the size and, consequently, with the cost of the precipitator. Most designs, therefore, are for efficiencies between 90 and 99.9 per cent.

The principles of precipitation are based on the fact that a gas stream, and particles suspended in it, become charged or ionized when passed between electrodes producing a highly stressed electrical field. The familiar corona discharge is evidence that this ionization is taking place. Gas molecules are normally balanced electrically, being made up of a nucleus having a positive charge and a number of electrons having negative charges which total and balance the positive charge of the nucleus. Ionization is the phenomenon of separating electrons from the positive nuclei. The gas molecules, or what remains of them, are now charged positively and are called positive "ions"; the negative electrons may remain free, or they may become attached to neutral molecules and so make up negative ions.

The ions and electrons become attached to the suspended particles, and the resulting agglomerates move across the electric field because of the force exerted upon the unbalanced electric charges they carry. Particles, no matter how small or how finely divided when thus charged, will move toward one or the other of the electrodes, depending on the sign of the charge and the position of the particles with relation to the electrodes. They will usually move toward the electrode having the largest surface, which is also usually the positively charged electrode. It is probable, however, that precipitation is more largely due to ionization of the gases than to induction.

In apparatus for the electrical precipitation of suspended particles, one of the electrodes is usually filamentary, or has sharp points or edges, and the other electrode has a relatively smooth and extending surface. It is possible, by impressing a high voltage across the electrodes, to obtain ionization of the gas and any liquid or solid particles suspended therein.

For electrical precipitation work, direct current is found to give much better results than alternating current, and the best results are secured when the ionizing, or so-called discharge electrode, is of negative polarity. It is possible to make the electric field around the ionizing electrode intense which tends to keep particles from being deposited upon it, thus moving practically all the suspended material over to and depositing it on the smooth or collecting electrode. The electric field adjacent to this electrode will be relatively weak because of its extended surface. This characteristic of the particles to migrate toward the weakest part of the field may be explained in part by the phenomenon commonly called "electric wind." The molecules of gases that have become ionized and the suspended particles to which electric charges are attached are propelled rapidly through the gases by the force of the electric field. The movement of these particles and molecules has an aspirating effect on adjacent gas molecules, setting them in motion—the direction being from the strongest to the weakest part of the field. Charges of both positive and negative sign are present in the gases, and they tend to move in opposite directions, but those which tend to move countercurrent to the electric wind have to overcome considerable resistance and are probably swept back or neutralized, especially if these charges are endeavoring to take with them suspended particles. By making the small electrode of negative polarity, the movement of the gases is set up by electrons and negative ions, which have a higher velocity than positive ions, and the results described above are then the most pronounced and most satisfactory. Even the minute particles (smaller than 1 micron) that are formed when metals are volatilized and condensed are ionized in this manner, and as they migrate through the gas they tend to come together and agglomerate into larger particles which are more easily transported into the hopper or receiving system.

Two groups of equipment are essential for these processes: electrical equipment to generate the high-potential unidirectional current required, and the precipitator in which the gases are cleaned.

To produce the high-voltage unidirectional current, it is necessary to have certain special electrical equipment. The usual commercial power supply of 220 or 440 volts, one or three phase, 25 or 60 cycle, alternating current may be used for the operation. Low-voltage control and regulating devices are placed in the primary circuit of a single-phase transformer, which is used to step this voltage up to the proper potential for the electrodes. A mechanical rectifier, synchronously driven from the same source, rectifies the high-voltage current from the special transformer before it goes to the electrodes. Where direct current only is available, it is necessary to use a motor generator set to obtain a low-voltage alternating current which can be properly transformed and rectified. A high-vacuum-tube rectifier of the type known as the Kenetron may be substituted for the mechanical rectifier.

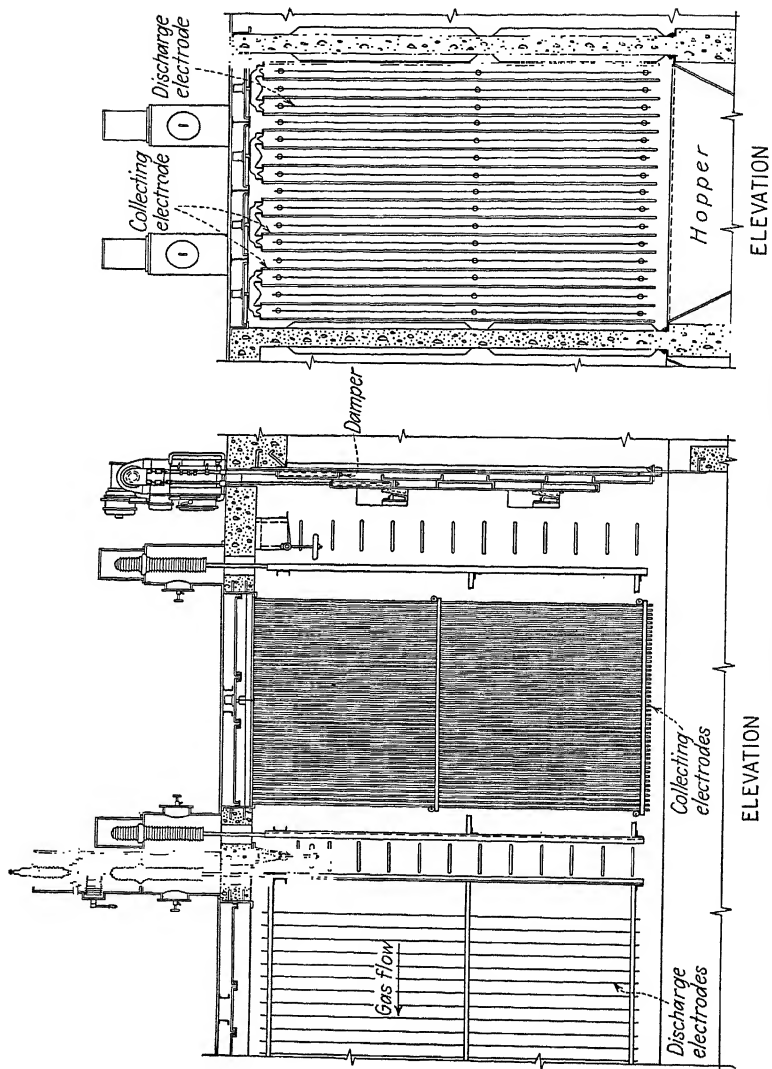


Fig. 6.—Cottrell precipitator. (Western Precipitation Co.).

The precipitator design depends on the type of gas treated and material to be collected, and also on the state in which the final collected material is to be handled, *i.e.*, whether it is to be dry or wet.

All precipitators have a shell or housing which should conform to the rest of the flue system in the nature of its general construction. This shell should be equipped

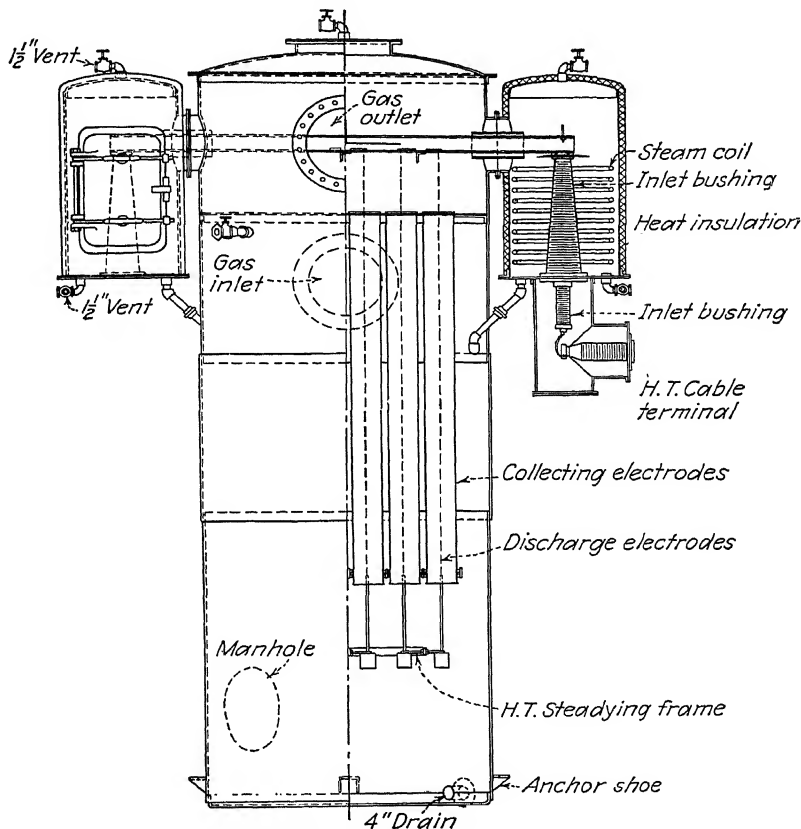


FIG. 7.—Sectional elevation, Cottrell precipitator.

with suitable hoppers or other proper means of handling the collected material. The variations will lie more particularly in the nature and arrangement of the electrodes themselves.

If dry material is to be collected, and if it is desired to have it dry when delivered, the electrodes will take one or the other of two usual forms. First, they may consist of a series of vertical tubes usually 6 to 10 in. in diameter, electrically grounded to the shell, and so arranged in it that the dust-laden gas must pass through them. These tubes form the positive electrode system which is grounded.

Each of the tubes will have at its axis a fine wire or similar electrode properly insulated from the tubes and shell and which, when electrically energized, becomes the charging or ionizing electrode.

Or, second, the electrodes may be a series of vertical "plates," with the ionizing electrodes suspended between them and so arranged that the gas must pass between the oppositely charged electrodes. Gas flow may be either vertical or horizontal. The plates may take one of several forms; they may be flat, they may be corrugated, they may be divided into a series of closely placed members, such as rods, or angles, or "pockets," so arranged that they form virtually a continuous curtain.

In all these arrangements where the material is to be recovered dry, provision must be made for the removal of the deposit from the electrodes and its safe conduct into a quiescent zone, usually a hopper, out of the main gas stream. This cleaning means usually takes the form of a rapping or a jarring system attached to the electrodes, but may consist of scrapers.

If the material to be collected is wet or if it can best be removed wet, then the electrodes may take a form similar to the ones first described above, but should receive special consideration as to corrosion resistance. The deposited material is usually removed by flushing the electrodes either continuously or intermittently with some liquid that is allowed to flow over the collecting surface.

When the particle-laden gas enters the space between the electrodes, the particles become charged and are moved to the electrodes in a volume that decreases logarithmically as the gas moves along the electrodes and as its dust burden is diminished. The fewer particles there are present, the fewer will be precipitated, by a constant percentage.

The size of the precipitator is determined by the volume of gases to be cleaned per unit of time, the properties of the material that is to be removed from the gases, the degree or percentage of cleaning desired, and, to some extent, by the temperature and composition of the gases themselves. Calculation to determine the size of the precipitator to be used may be based on the following equation:

a = gas volume to be treated per unit of time, usually cubic feet per minute.

b = function of equipment and maintenance cost.

c = value of solids treated.

d = a function of the specific precipitation rate.

x = precipitator size.

$$x = a \left(\log \frac{b}{c \log d} \right)$$

When the size of the precipitator has been determined, the capacity or size of the electrical equipment may then be determined for the Cottrell electrical-precipitation processes.

These processes have been applied to various problems for the removal of suspended particles from the gases, and the diversity of their application to both dry and wet-gas cleaning is shown by the following summary:

1. Nonferrous metallurgical fume and dust: recovery of copper, lead, tin, zinc, silver, gold, bismuth, arsenic, and potash, etc. *Operations:* sintering machines, reverberatory furnaces, blast furnaces, converters, roasters, driers, etc.
2. Acid problems:
 - a. Sulphuric acid concentrators (new, spent, and sludge acid), parting kettles.
 - b. Nitric acid: nitrating operations. Phosphoric acid: phosphate-rock reduction. Hydrochloric acid: roasting operations, pickling, etc.

- c. Miscellaneous: recovery of chlorine, bromine, etc.
3. Smoke, soot, and cinder:
 - a. Boiler-plant gases.
 - b. Engine roundhouses.
 - c. Lampblack from oil. Carbon black from natural gas.
4. Cleaning of gases containing sulphur dioxide:
 - a. Hot gas (pyrite burners): sulphuric acid manufacture.
 - b. Cold gas (pyrite burners): sulphuric acid manufacture; (sulphur burners); sulphite liquor production.
5. Combustible gas cleaning:
 - a. Iron blast-furnace gas cleaning.
 - b. Hot producer-gas cleaning.
 - c. Tar recovery from producers, illuminating gas, coke ovens, low-temperature (fractionation), wood distillation.
6. Air cleaning:
 - a. Small-capacity installations: foundries, machine rooms, etc.
 - b. Large-volume installations: slate crushing, etc.
7. Miscellaneous applications:
 - a. Dust and fume from industrial operations:
 - (1) Potash from cement kilns.
 - (2) Large electric furnaces.
 - b. Dust and fume from miscellaneous chemical-plant operations.
 - c. Oil industry: catalytic cracking units for recovery of catalyst.
 - d. Organic materials: spray drying, milk, fruit juices, etc.

The following table gives some typical figures on first cost, together with operating costs, of a representative group of Cottrell installations:

	Oil re- finery crack- ing catalyst	Metal- lurgi- cal fume	mist con- cen- tration	H ₂ SO ₄ mist SO ₂ gases	Boiler house gases (pow- dered coal)	Zinc roaster gases	Pro- ducer gases (detar- ring)	Spray drying solids re- covery
Investment per rated cubic foot per minute.....	\$2.25	\$0.56	\$1.80	\$3.75	\$0.11	\$3.00	\$1.28	\$0.50
Temperature of gases treated, °F.	500	500	250	70	300	1200	100	150
Cleaning cost per 100,000 cu. ft. per hr.	{ \$0.05 \$0.10 \$0.05	\$0.09 \$0.10	\$0.18	\$0.01	\$0.16	\$0.11	\$0.05
Power consumption, kw.-hr. per 100,000 cu. ft. per hr.	0.25	0.75	1.00	0.67	0.25	0.55	0.88	0.56
Efficiency of removal of sus- pended matter, per cent.	90.6	90-95		99.9+	90	95	95-98	95
Length of discharge electrode, ft. per 100,000 cu. ft. per hr. treated.	450	360		150	170	280	100	280
Spacing plate to plate, in. (or pipe diameter).	8-11	6	diam.	8 diam.	4		6 diam.	4

Cottrell apparatus has been designed and is operating on gases at temperatures up to 1200 to 1300°F., and the apparatus is so designed that the temperature drop through it is approximately 100°F. Here a twofold problem is solved, *viz.*, gas cleaning and heat conservation. Cottrell installations operate continuously and, in larger installations, provision is made for by-passing any of the precipitator units for cleaning or repairs. Maintenance of these installations is usually figured at 1 to 2 per cent of the first cost, the apparatus in any given case being designed to suit operating conditions, *e.g.*, the use of special materials to obviate corrosion caused by high temperatures or chemical action.

The essential data in any gas-cleaning problem are as follows: (1) volume of gases to be treated per unit of time; (2) temperature of such gases, with a graph showing maximum and minimum as well as usual working average; (3) pressure of such gases with information on related operations and the effect of resistance to gas flow in such operations: furnaces, ventilations, fans, flues, stacks, etc.; (4) humidity of such gases, relative humidity and dew point for gases under 212°F., grains per cubic foot in gases above 212°F.; (5) gas composition, particularly acid constituents; (6) quantity of suspended matter per cubic foot of gas; (7) physical and chemical properties of such suspended matter; (8) degree of cleanness required (efficiency); (9) availability of power and water; (10) utilization of cleaned gases and recovery of by-products; (11) operating costs.

CHAPTER XIII

BRIQUETTING APPLIED TO METALLURGY¹

Definition.—Briquetting is the process of fabricating, through the medium of a press, with or without other operations in connection therewith, fine materials into blocks of larger size but essentially the same composition, for a purpose involving the destruction of the blocks as such, either by direct useful consumption or as a step in a melting or reducing operation.

In metallurgy the following are dealt with: (1) ores, flue dusts, and other by-products that are subject to smelting; (2) metal scrap, which is subjected to melting; and (3) the fuels and fluxes with which that smelting or melting is performed. All these raw materials may be subjected to briquetting, under certain conditions, to the betterment of the process. Fuel briquettes have been used in smelting and metal melting in comparatively few instances. They have been used in stationary power practice to some extent, to a greater extent on locomotives, and very widely as domestic fuel. In the field of ore smelting, briquetting has had a wide application. In general, where air must be driven through a metallurgical charge, the substitution of briquettes for fine materials provides a free passage of the blast. The products of the iron furnaces—flue and filter dust—and, to a smaller extent, of the blast furnaces involved in copper and lead smelting have been briquetted for many years, and recently a close and intensive study has been given to improvement of the practice. In the metallurgy of zinc, where recovery depends upon the vaporization of the metal content with subsequent condensation, the process has been facilitated by briquetting the entire ore charge. Fluxes are seldom briquetted by themselves, but the practice of briquetting ores and by-products with their fluxes has been found advantageous in some cases.

In general, metal scrap is briquetted without binder. The by-products of the furnaces are briquetted by the addition of a corrosive substance which forms a binder by chemical action or with such reagent with the assistance of a binder. Fuels in nearly all cases require a binding medium to form good briquettes. There are exceptions. Milk of lime is sometimes added to assist the bond in metal-scrap briquetting. Inert ores which do not contain elements that will react to chemical reagents require a binder, whereas, in the fuel class, sawdust, peat, and the braunkohle lignites are briquetted without the addition of any binding medium.

Briquetting of Metal Scrap.—There is a very large production of metal turnings and chips, known generally as swarf or light scrap. The heavy scrap of all metals brings higher prices in the open market than does the swarf, for several reasons: The transportation and handling of tangles of light scrap, with a large volume in proportion to the weight, are expensive. Charging is difficult. During the melting excessive oxidation occurs, and with the volatile metals, a large loss by volatilization occurs.

When such a charge is briquetted, a bulky and unwieldy mass is transformed into uniform dense blocks, ranging in quality from superiority over heavy scrap of the same metal, as with steel for open hearth, to a near equality, as exemplified by cast iron and yellow brass. In all cases, handling and transportation are much facilitated, melting time is reduced, and the quantity of metal per heat increased.

¹ Original chapter by Albert L. Stillman, revised by the editor, 1943.

Where the turnings or chips are small in size, dry, unmixed with heavy scrap, and of a metal not too hard and resistant, the material can go directly to the press. Usually, one or more conditions are present, which necessitate some preparation before pressing can take place. Heavy scrap, caught in the molds of the press, may cause scoring and breakage, and certainly its presence causes lines of weakness in the briquettes. Long turnings rebel against packing into the receiving hopper, resulting in very small briquettes, with consequent high cost of operation. Hard material may call for a heat-softening or annealing treatment, usually too costly for steel- and iron-briquetting practice, but most useful for brass. Oil should be removed, and deoiling is accomplished by burning off the oil in the same operation that softens the metal, or by removal of the oil or grease by a solvent, followed by centrifuging. This, too, is usually confined to nonferrous practice.

In the briquetting of metal swarf for melting, the following features of preparatory treatment may be cited:

1. *Shredding* the tangled mass of turnings—performed by a tedder, consisting of a rotor with projections operating against a stationary plate with retarding members.

2. *Pneumatic separation* of the light from accompanying heavy scrap—nuts, defective “work,” and the like. The materials are dropped vertically in a thin stream before an air blast, which deflects the light turnings into an air chamber, at the bottom of which is a conveyer. The heavy material drops to a cross-conveyer below and is used as heavy scrap, having no future part in the briquetting process.

3. *Cutting* the long turnings into shorter lengths. Several cutters have been designed in the United States for breaking up long turnings. This part of the operation was long a bar to profitable working, but there are now several cutters in the market operating efficiently and at low cost.

4. *Drying, Deoiling, and Annealing.*—The removal of oil and water and the softening of the metal are performed in the same apparatus, a revolving rotary drum. The feed end is a little higher than the discharge end. A burner, located at the discharge end, throws a flame into the furnace, exhausting through a small stack at the feed end. The chips are fed from an elevator into the hopper. As the drum revolves at a moderate speed, the metal is carried to the discharge end partly by gravity and partly by vanes inside the furnace, so shaped as to push the metal along in one direction. The dried chips discharge automatically through an opening in the shell, which comes into the discharge position with each revolution. However, as already indicated, removal of oils and greases may also be done by solvents.

Usually this work is confined to the copper and nickel alloys, and oil in the ferrous metals is allowed to remain in the briquettes, not being especially objectionable in the open hearth, and the cost of the treatment being higher than the selling price of steel or iron briquettes would justify. In practically all briquetting operations, however, serving material to the press hot, within limits, contributes much to the ease of operation of the press and to the strength and quality of the briquettes produced.

The briquetting of scrap at high temperatures originated in Germany, but W. Johnson & Son, Ltd., of Leeds, England, has done a great deal to develop this process, and it is understood that some of their installations are briquetting steel and iron scrap at red heat or close to it, using a tamping hammer that delivers 250 blows per minute.

It is understood that some English companies are heating aluminum and magnesium scrap almost to its melting point and are then extruding it as a substitute for briquetting.

In the treatment of nonferrous scrap such as aluminum or brass, the materials should be put through a magnetic separator before briquetting. With all possible

care, there is an iron pickup in the treatment of nonferrous scrap which is most undesirable.

5. *Pressing*.—The usual method of briquetting metal chips is to apply a very heavy pressure and to rely on the pressure to break down the elasticity and resistance of the metal in the block formed. No binder is required in the majority of instances. Cases are known where the addition of a small proportion of milk of lime assists the bond very much. Fresh water—and, at times, salt water—is added to the scrap of the various metals to induce a slight corrosion, forming a binder of ferric oxide. Later, acid sulphite liquor has been advocated in the briquetting of cast-iron borings, as the acid contained in the sulphite, acting upon the borings in conjunction with the colloidal pectin in the sulphite, produces an excellent cement. So quick is its action that roll-press practice is permissible, with the consequent lowering of pressures and raising of possible tonnage many times over. In the bulk of existing practice, however, especially in the United States, hydraulic practice is used. In Great Britain, toggle presses having lower pressures have been used to advantage.

Where high pressures are used, and no binder or corrosive action is employed, the bond is secured from one of two phenomena, or, perhaps, a combination of both. Where comparatively soft metals are used, as, for instance, lead, the metal actually flows under pressure and all interstices between the chips are thereby filled. Where hard brasses are briquetted there is but little flow, but a very considerable skin tension, accompanied by cracks, which interlock on the tangent surfaces, forming a friction bond. As the ultimate resistance of the metal is passed by the working pressure, the interlocking is permanent and the bond is secure.

Four large and two small hydraulic presses have been in common use in the United States for the briquetting of metal swarf. The large are: (1) the Ronay-Gilmore type, an improved design of the Ronay press used in Germany; (2) the Duryea type; (3) the Jacomini type; (4) the Timken type. The smaller sizes are the Stevenson-Little and the Gilmore.

The Ronay-Gilmore Press.—The press used in the United States is a three-post construction. The two-post, lighter, is used in Europe.

The rigid frame of this press is divided into upper and lower platens, supported by the columns. Six molds are carried in a horizontal rotating table. Either 5- or 6-in. molds are used, depending on the unit pressure desired. To the upper platen are attached cylinders carrying the packing, ejecting, and table-turning rams. The pressing position is between the packing and ejecting positions, forming the apex of the operating triangle. The pressure ram operates from below, against a counter-plunger in a cylinder above. Three of the molds are subject at the same time to the various operations of packing, pressing, and ejecting, while the alternate three molds are idle, though two of them are carrying material—one, swarf packed but not pressed, and the other, a briquette prior to ejection. On the completion of the ram action in each case, the table-turning mechanism comes into play, bringing the alternate trio of molds into the working position. At the pressure position the pressure is applied in two stages, the accumulator pressure entering first and considerably reducing the volume of chips in the mold. At the ejection position a descending ram comes into play, driving the briquette downward through a chute into the receiving hopper.

It will be noted that there is a counterplunger so that pressure can be applied to both ends of the briquette. This is most important particularly in the treatment of iron and steel scrap. In any pressing operation in which the pressure acts against a rigid mold back, the briquettes are better formed at the ram end and are quite badly compressed at the fixed end of the mold.

Briquettes made by this press are 5 in. in diameter where a pressure of 36,000 lb. per sq. in. is required, and 6 in. in diameter where a working pressure of 22,000 lb.

is sufficient. The briquettes are usually about 5 in. high for steel borings, and weigh between 16 and 20 lb.

The Duryea Metal-briquetting Press.—The mold is stationary, supported in a frame of four columns. The mold is fed by means of an automatic plunger and the various operations are brought into play, accurately timed by a camshaft on which six cams are mounted. The mold is filled by a charge carrier, and hydraulic pressure forces a ram downward upon it. The hydraulic pressure is little more than a packing pressure, and, as soon as it is complete, the pressure from an internal-combustion engine's

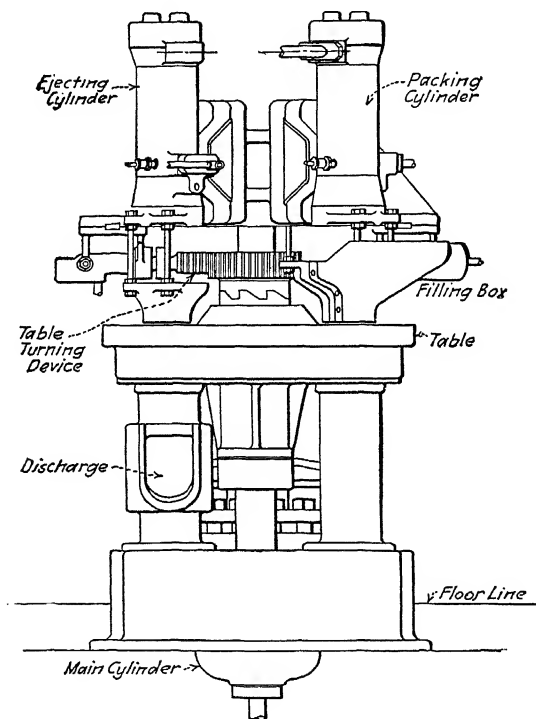


FIG. 1.—Ronay-Gilmore press.

explosive charge is brought to bear upon another ram, operating from above, which is forced downward upon the hydraulic fluid. The pressure thus transfers to the pressure ram, which is forced downward, delivering the force of the explosion to the charge in the mold. The pressure developed is extremely great, and the briquettes formed are very compact. After the action of the power-cylinder exhaust valve is opened, and the power piston forced downward, the opening of the hydraulic valve then following relieves the pressure ram from its pressure and it rises clear of the mold. The counterplunger at the base of the mold rises, putting the briquette in position for ejection by the coming new charge. The briquettes are small, cylindrically shaped, with rounded ends.

This press has been used for the briquetting of cast iron, brass, and gas-mask charcoal.

The Jacomini Press.—Filled molds are placed successively under a ram and removed as the operation is finished. A ram is moved downward on the charge by hydraulic pressure. The press is supported by four columns. After the maximum of hydraulic pressure has been reached, live steam is admitted to the operating cylinder. The structure of the cylinder and the ram head is such that for an instant the resistance to the plunger practically disappears, so that the descent of the ram on the charge has the effect of a hammer blow.

After the completion of the pressure the ram is raised, the mold containing the briquette is removed, and a new mold is placed in working position. The briquettes are 4 in. in diameter, about 5 in. wide, and weigh about 10 lb. The work of this press so far has been confined to cast-iron borings.

The Timken Press.—This press was named from its installation at the Timken Roller Bearing plant, and is made by the Southwark Machine Co. In it, a horizontal piston works against a retreating mold and counterplunger. The pressure application is mechanical.

The Stevenson-Little Press.—This press consists of a mold mounted on springs with the pressure cylinder and piston above. The mold is filled through a conical hopper through which the plunger descends. As the pressure is exerted, the springs are compressed and the mold lowered. When the pressure is released, the mold assumes its regular position. A hand lever moves the block, carrying the counterplunger to the back of the press, giving room for the ejection of the briquette on the downward movement of the ram.

The Gilmore Horizontal Press.—This press is the simplest in design of the hydraulic class of briquetting presses. A piston, hopper, and mold lie in a straight, horizontal line. After the hopper is filled with the charge, the piston is moved forward, packing the turnings. Then the pressure is built up to maximum and the briquette formed. The pressure is released and the mold raised by means of a hand lever, so that the briquette can be ejected by the piston, again advanced.

Steel briquettes are ideal for charging into the openhearth furnace. Their regular shape is of help in loading the charging box and permits larger loads per box. Charging time is saved. In the melting they immerse rapidly in the bath, melting from the outside. The melting loss is small. In general, they may be considered as superior to heavy melting steel for this purpose.

The briquetting of cast-iron borings is undoubtedly the cleanest and best method of preparation for melting down such borings in cupola practice. Those who have tried all methods prefer the briquettes to the boxing of borings or melting them down in cans or pipe. Briquettes are not the equal of heavy scrap in cupola practice, however, nor can a 100 per cent briquette charge be used unless a low grade of iron is permissible. Where too great a proportion of briquettes is used in the charge, the iron takes on the characteristics of white iron. In no metal briquettes are the voids absolutely eliminated, and 75 to 80 per cent metal constitutes excellent briquetting practice. Consequently, in the upper zones of the furnace a large proportion of graphite and silicon is burned up and the briquettes are likely to take up more sulphur than the ordinary metal charge. Where small proportions are used—10 to 20 per cent of the charge—excellent iron is made.

The briquetting of copper alloys is especially important, and the importance increases with the percentage of volatile metal, such as zinc, incorporated in the alloy. The melting down in crucible furnaces of light swarf and fine turnings increases by surface contact the tendency of the zinc to escape. The same turnings in briquette form behave very much as do ingots. The advantages of handling, of saving of melt-

ing time, and prevention of oxidation are also present. Oil should always be removed, preferably by burning out prior to the briquetting. If the oil is retained, the lubrication between the particles of swarf weakens the bond. Again, the annealing and the softening of the metal increase the efficiency of the pressure applied. Practically all nonferrous scrap contains particles of iron or steel which must be removed by magnetic separation, as already mentioned. If oil is not removed, the work of the magnetic separator is largely negative and a proportion of iron remains in the scrap to the great detriment of the resulting ingot. Dry turnings, if well cut up, are readily separated by the magnet.

Briquetting Aluminum Chips.—It is strange that more has not been done in the briquetting of aluminum, as in no case would it seem that briquetting is so necessary. Aluminum oxidizes very easily, and in the melting of small-size chips a great amount of oxide is formed, which remains with the metal to its great disadvantage. By briquetting, the contact of the surfaces with the air is reduced, and the briquettes are readily charged and submerged. The use of extruding devices for the preparation of aluminum and magnesium chips has already been commented on. A recent patent (U.S. patent 2332277, Oct. 19, 1943, to Max Stern) covers the treatment of fine material by pressing at 250 to 500°F. after heating in a reducing atmosphere to clean the material.

General Considerations.—A survey made by the editors of *Iron Age*¹ of 12 large American plants showed the unanimous opinion that briquetting of metal scrap paid. The advantages were that metal losses in melting were lower; there was greater ease of charging the material to the furnace; it was easier to handle briquettes in and out of storage; the storage space was less than that required by the nonbriquetted material; and a closer control of the melts was possible.

There was little similarity in their operations. The capacities of the 12 plants varied from 7 to 35 tons of finished briquettes per 8-hr. day. The diameters of the briquettes ranged from 2¾ to 5 in.; the briquettes varied in thickness from 2½ to 4 in.; and their weights varied from 2¼ to 14 lb. The wide divergences in operations render the unanimous opinion in favor of briquetting the more striking.

For a plant turning out 3 to 5 tons per 8 hr. of briquettes from light scrap and 5 to 10 tons per 8 hr. from heavy scrap, the pulverizer will require 30 to 50 hp., the magnetic separator 1 to 2 hp., elevators about 2 hp. each, the feed table to the press 1 hp., and the briquetting press itself 10 to 15 hp. It is estimated that the cost of briquetting will be about \$1.40 per ton and that the direct savings will be about \$4 per ton.²

Bundling or cabbaging, whereby long strips or sheets of metal are pressed into rectangular bundles suitable for melting, has been established practice for some time. It is a recognized economy, and bundles are quoted at an advance over loose scrap. Bundling, as such, is limited to the long flexible turnings and sheets. Nothing can be done, so far as these cabbaging operations are concerned, with the loose fine needles, borings, punchings, and brass washings. That is purely a function of metal briquetting. In all metals a distinction should be made between bundling and briquetting. It must be emphasized that the briquette and the bundle are in no sense competitive, as they use material in entirely different forms.

Methods of Securing Bond.—The problem of first importance in the briquetting of fine materials is the securing of a bond between the particles. Different methods must be used for different materials. The following methods are in use. 1. By pressure only. (a) The applied pressure brings about a restricted flow of the material so that the particles interlock and fill the interstices between them. Examples:

¹ *Iron Age*, Vol. 150, p. 31, Oct. 29, 1942.

² *Engineering*, Vol. 152, p. 306, Oct. 17, 1941; Nov. 7, 1941, p. 366; Nov. 28, 1941, p. 426.

chips of babbitt metal, lead, or other soft metals. (b) By skin tension of adjacent surfaces. Examples: small brass chips, metal borings. (c) Interlocking of particles under pressure. This is usually combined with skin tension or flow. Examples: long turnings, as steel and hard nonferrous metals.

2. The material is made self-binding in whole or in part by processing or chemical action. Example: cement and bricks, wherein the entire material is processed by the use of heat or by the addition of water until it becomes self-binding. Materials used for briquetting purposes are seldom of this nature. The following examples may be cited of materials whose briquetting is best accomplished by rendering part of the material self-cementing. (a) By hydrolized cellulose, usually combined with bitumen—peat, braunkohle, and lignite. (b) By chemical action, wherein a reagent is used to make part of the material self-cementing. Example: blast-furnace flue dust, zinc ores, and dusts.

3. Materials where binder is extracted during a preparation process. Example: fine wood waste, where rosin or gum is extracted from the particles by preliminary steaming; bituminous coals subjected to heating and mastication, whereby the contained bituminous matter assists an added binder in maintaining bond; bituminous coals subjected to briquetting pressure and heat at the same time, whereby a binding tar is evolved and sets.

4. Materials in themselves inert briquetted by added binders. Of the binders that may be added for this purpose the following are the more prominent: (a) Inorganic: lime, magnesia, gypsum, Portland cement, slag cement, clay, water glass. (b) Organic, insoluble in water: wood tar, gas tar, pitches from the above tars, natural and petroleum asphalts. Organic water-soluble: glue, starch, gluten, sulphite liquor, molasses, and other similar compounds.

Briquette Plant Design.—Where binders are added or are present in the material or are made by processing the material, an equal distribution of that binder throughout the mass is highly necessary. The methods used to secure such distribution are very similar in all materials, so in this respect the briquetting of fuels and metallurgical products and ores are closely allied. Most added binders are liquid, but some pitches are used in solid form. None the less, the principle of equable distribution applies in all cases, and argument against the use of solid binders is founded on the difficulty of distributing them through the mass. The preparatory treatment is, therefore, of at least equal importance to the pressing mechanism.

In the design of a briquetting plant, the character of material and the character of the bond and their relation to each other must be constantly kept in mind. The material itself should be sized properly and should consist of a mixture of sizes beginning with a determined maximum. It has been found in the case of anthracite coal that No. 3 buckwheat is the largest size compatible with the best work, and the mass should contain a large proportion of smaller sizes to fill the interstices; otherwise an excess of binder must be used. Similarly, if a large proportion of fine material occurs, an excess of binder must be used in order that each particle be thoroughly coated. The problem of the preparatory treatment is so to coat or treat each of the various-sized particles of material involved with natural or added binder that they will mutually adhere, and that, too, with the smallest possible expenditure of material and effort.

It is usual—and necessary where an excess of moisture is contained in the material—that it be dried prior to mixing. The heat tends to preserve the liquidity of the binder added later on, if a hydrocarbon binder be used, and thereby to assist in distribution. Where it is necessary that low temperatures in drying be used—as with lignite, where the danger of explosion is present—steam drying is ideal. In most cases, as in the briquetting of anthracite, open-fire driers are best and lowest in cost. Some-

times, where bituminous coal is briquetted, a semidirect type of drier proves efficient and safe. Where flue dusts are briquetted, the addition of the reagent usually produces a palpable increase in temperature. In fact, as the hot dust as received from the furnaces is the easiest to handle, there is seldom a drying problem in connection therewith. At all events, if the lignite and the Zwoyer process for coal are excepted, there is nothing in the drying of materials for briquetting which is of special nature. The steam drying of lignite is performed either on steam tables or in tubular revolving drums, the steam being carried in the drum and the lignite in the tubes. In the

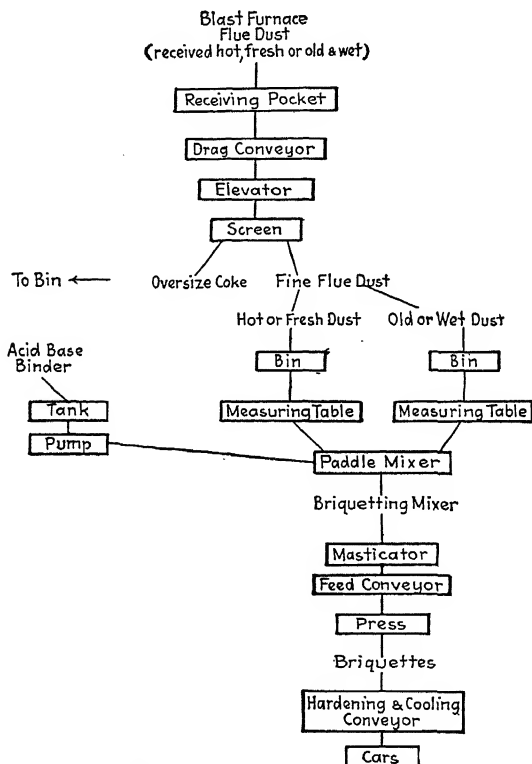


FIG. 2.—Briquetting plant flow sheet, drying apparatus not shown.

Zwoyer process for coal briquetting, drying is simply and easily effected by passing hot gases through a pugmill against the travel of the material and prior to the addition of binder.

From the driers the material passes into pugmills or distributing screws. If binder or reagent is added, the pugmill is preferable. If, as in the lignite or braunkohle industry in Germany, the binder is contained in the material, the distributing screw is well adapted to the situation. At the head of the paddle mixer or pugmill the stream of binder or reagent is added. If binders such as asphalt are used, it is well to

use steam jets throughout the length of the paddle mixer. If water-soluble binders are used, care must be taken that the fine material is not overheated in the drier lest there be undue evaporation of water and consequent difficulty in later mixing. The condition of the material on emerging from the paddle mixer or series of paddle mixers should be damp and homogeneous. In the past it has been the practice to pass material from the mixer directly to the press, with possibly a passage through a vertical fluxer intervening. Especially is this true today in European practice, where solid pitch is ground together with coal, run through a heated paddle mixer and fluxer, and then pressed into briquettes. In American practice, an additional treatment to secure excellence of product has been found beneficial. A prominent treatment involves the use of a "masticator," whereby the mix from the pugmill is ground for a time under heavy rollers moving about a circular pan. The feed and discharge

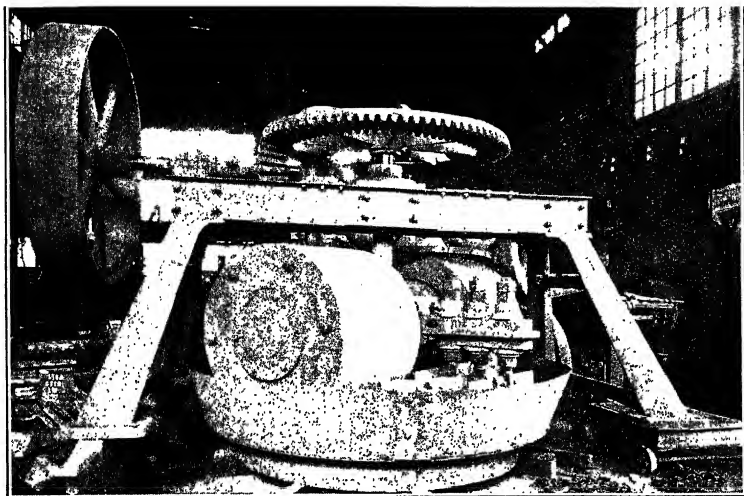


FIG. 3.—Masticator for preparing briquetting mixtures.

of such a machine is continuous. The product from the masticator is comparatively dry as compared with the material that has entered, but the distribution of the binder or reagent and its cementing quality has been greatly improved. The binder is not only around the particles but penetrates them. Where chemical action is necessary to make the binder—as in blast-furnace flue dust—the rapidity of the reaction is much increased by mastication.

Sometimes, and notably when tars, pitches, or asphalts are used as the binder, a fluxing or tempering apparatus is placed in the system at this point to ensure an equable press feed. In fact, many presses are made with vertical fluxers attached. In this apparatus the material is passed down a cylinder in which is contained a vertical shaft carrying shelves. The material passes down from shelf to shelf in an atmosphere of steam and is then in excellent condition as a feed to the Belgian or piston presses. Frequently, the cooling of the binder, especially during mastication, results in the formation of lumps, which are broken up by the subsequent tempering treatment. A paddle mixer or screw conveyer with steam jets may be used to perform a

similar function. Where water-soluble cellulose binders are used, this work is hardly necessary and the mix is conveyed to the press. Where the modern rotary press has been installed, designed to take a gravity feed, obviating the use of rotating feeding devices, again there is little need for such a device, for such a press is capable of making briquettes regardless of the lumpy condition of the feed.

Two types of briquetting press are in common use: (1) mold-and-plunger type; (2) roll type.

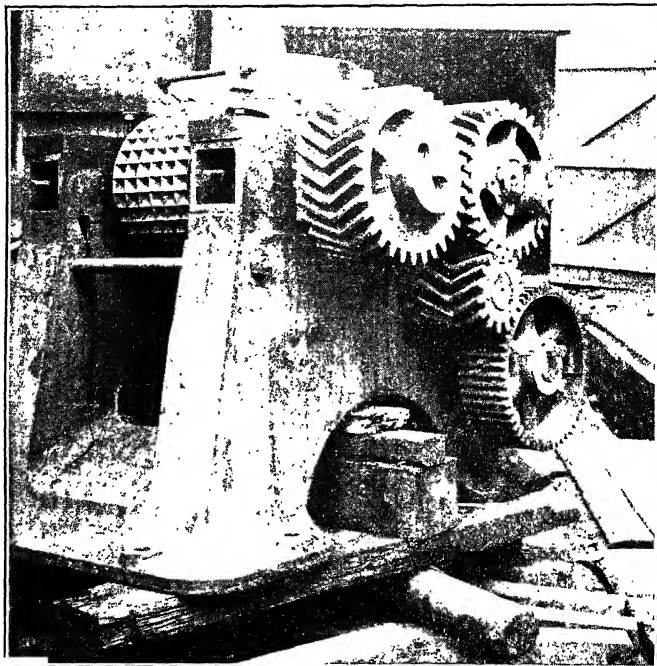


FIG. 4.—Zwoyer universal roll-press for briquette making.

In Europe, all metallurgical briquetting and a large part of the coal briquetting is done by piston-and-mold methods. The piston-and-mold presses may be roughly divided into four classes: (1) closed-mold horizontal-table type; (2) closed-mold vertical-table type; (3) reciprocating-mold vertical type; (4) open-mold horizontal type, used in braunkohle or lignite briquetting. Types of each one of these presses will be found described at the end of the chapter. There are various types of the roll press as well, but the differences are chiefly in the method of feeding. The two main types are: (1) the Belgian, or mechanical-feed type; (2) the gravity-feed type, such as the Zwoyer. Several presses combine in a measure the rotary-press with the piston-and-mold principles. Notable among these is the Rutledge.

As between the two main types of press the following considerations rule: The rotary presses make small briquettes, easily shoveled and quick in setting. The

piston presses require less binder for the formation of good briquettes but do require a larger outlay for labor and repairs. A high-tonnage rotary press is very much lower in first cost than a piston press of the same capacity. Where large bricks are wanted and where the scale of pay for labor is low, the piston press must be used—especially in the packing of coal briquettes in the bunkers of ships, and, as a consequence, most of the briquettes made in Wales for overseas are of this class.

On the Continent and in England, where there is a large demand for briquettes of the size and the weight of bricks, for combustion in tile stoves, the piston press is the

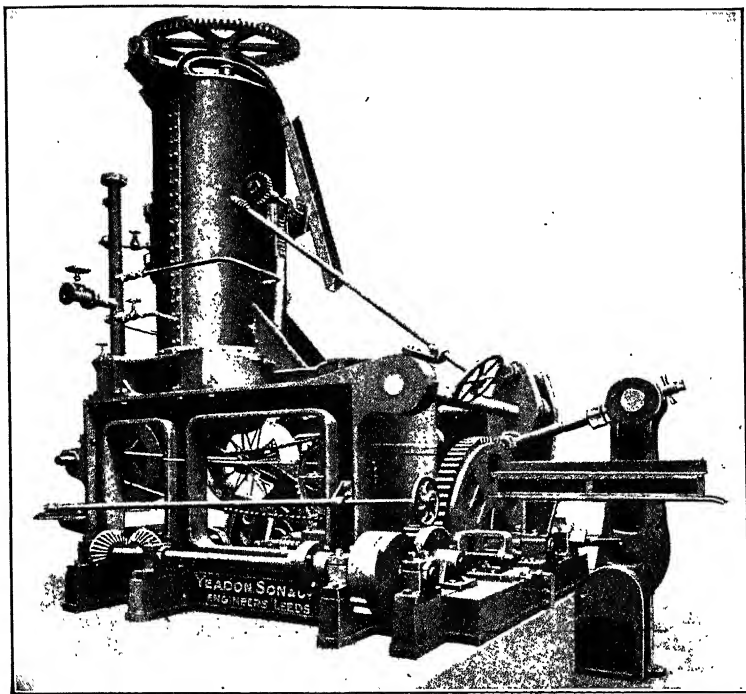


FIG. 5.—Revolver press—mold table rotates in vertical plane.

choice of the majority of establishments. In North America, where domestic heating is largely done by means of central furnaces, the shoveled briquette has met with the greater favor, to such an extent that there are practically no piston presses employed in fuel-briquette manufacture, except for those made by the Rutledge press in the Middle and Far West, largely for power purposes, and these rarely weigh more than 1 lb. each.

In metallurgical briquetting, involving the preparation of ores and flue dusts for smelting, for many years the practice has been to use one of the various types of piston presses. The examples of such presses as used in Germany are Schumacher, Gröndal, Tigler; in Great Britain and the Colonies, the Sutcliffe, Yeadon, Johnson. (The latter two, of the closed-mold vertical-table type—colloquially known as the “revol-

ver press"—have been very largely used.) The revolver presses, in fact, have proved successful in both coal and metallurgical briquetting. In the United States the White press has long held vogue, particularly for briquetting nonferrous flue dusts, while the Schumacher press has been employed in several instances for the briquetting of iron dust. At times, ordinary brick presses have been used. Latterly, the tendency has been to substitute the rotary press on account of the high tonnage obtained, the cheapness of operation, and the convenient size of the briquette. In general, in metallurgical operation small briquettes are more satisfactory than large. Presenting as they do a greater surface in proportion to their weight, they dry out in

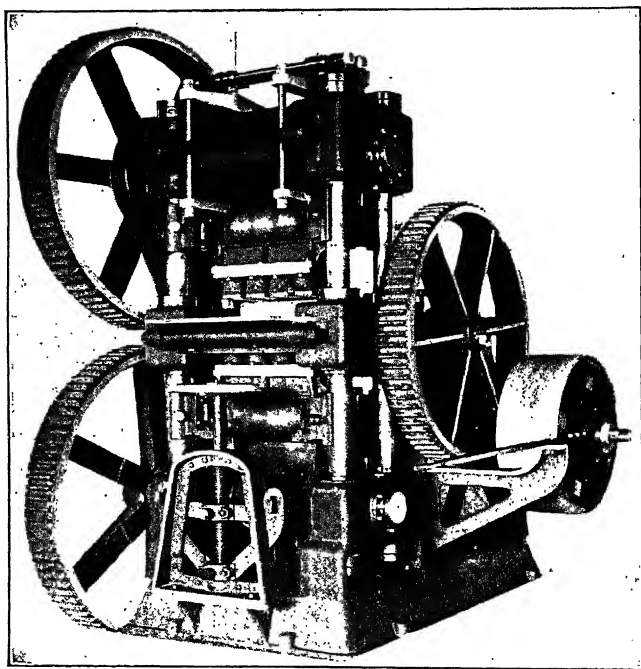


FIG. 6.—Fernholz geared press.

faster time, and thereby more quickly attain a permanent set. In the furnace, they lend themselves well to the making up of the charge and to even reduction. The New Jersey Zinc Co. has successfully employed overhead-feed Belgian presses in the manufacture of briquettes of zinc ores and anthracite for the production of zinc oxide by distillation. In the reclamation of iron flue dust, the roll press has obtained favor, largely as a result of the perfecting of the gravity-feed type of press. The gravity-feed type has been made possible by careful design, the rolls being properly proportioned to their requirements, so that they become self-feeding. The pockets take their load from a superposed hopper filled with the briquetting mix. Higher pressures can be obtained, as desired, by varying the head in the column—which is manifestly impossible where the feed is controlled by rotating fingers. In the briquetting of

metallurgical mixes, and particularly of the iron flue dusts, where the permanent set is in practically all cases obtained in the briquette about an hour after the material has passed through the press, it has been difficult to secure sufficient pressure in the rotary press to enable the newly formed briquette to fall from the machine. In the Belgian press the breakage has heretofore been prohibitive. In the gravity-feed type of roll press the breakage is little. Most briquettes require some treatment after leaving the press, even if it is cooling only.

Briquetting in the Magnesium Industry.—The rise of the Pidgeon process for magnesium has brought about quite a study of briquetting, for it was early found

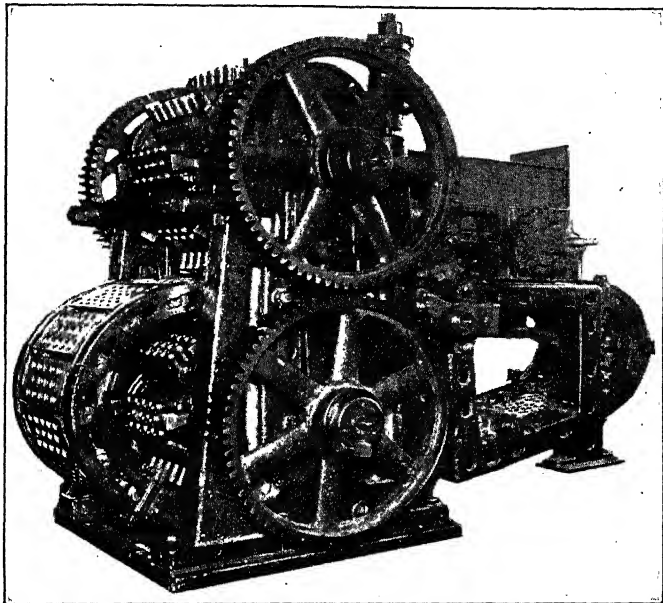


FIG. 7.—Rutledge press—combines rotary and piston pressing.

that thorough intermixture of the magnesite and ferrosilicon entering into this process was a necessity. The New Jersey Zinc Co. investigated this part of the process at some length and published the results as *Technical Publication 1707* of the American Institute of Mining Engineers.¹ The study of this problem by the New Jersey Zinc Co. is so illuminating that it is abstracted at length.

Pidgeon originally made his briquettes by mixing finely ground calcined dolomite and ferrosilicon, briquetting the dry mix in a plunger press under high pressure, though he planned to use a roll press in his commercial operations. The unsatisfactory results of early experiments by New Jersey Zinc on roll pressing led to experiments with wet briquetting.

Sufficient water was added to slake the lime of the calcined dolomite and to provide a plastic mix. The excess water was removed after briquetting by first heating to

¹ Pierce, Waring, and Fetterolf, *Some Developments in the Production of Magnesium from Dolomite by the Ferrosilicon Process*, February Meeting, 1944.

110°C., and the water of hydration was removed by heating to 600 or 700°C. in a muffle. The mixing was originally done in a pan mill with a heavy muller, but satisfactory results could also be obtained in a paddle mixer or a cement mixer. It was found advisable to densify the mix by passing it through rolls before briquetting. These rolls could be smooth, corrugated, or briquetting. The densified mix was then briquetted in rolls at a pressure of about 2000 lb. per sq. in. on the mix.

It was found that, by limiting the temperature of the heating gases, oxidation could be avoided and that by discharging briquettes at a temperature of 650 to 700°C. the CO₂ content was low, so that direct-fired preheating of the briquettes was possible in an internally fired rotary kiln, and it was also found that the strength of the briquettes was adequate to prevent serious breakage due to the tumbling motion of the kiln.

If the calcined dolomite was of the type in which only the calcium oxide hydrated, the density of preheated briquettes was about 1.7 g per cc.; if the dolomite was of the type where the magnesia hydrated, the density of the preheated briquettes was reduced to about 1.3 grams per cc. The expedient of hard burning the dolomite led to a slow hydration of the magnesia in the finished briquettes with constant cracking and disintegration. If the water added was only sufficient to hydrate the lime, the magnesia later picked up water from the air, resulting in disintegration of the briquettes.

Later in the research, a Komarek-Greaves press was obtained with which it was possible to obtain better results by dry briquetting than had been obtained theretofore.

The Komarek-Greaves press is a high-pressure roll press designed for briquetting dry materials. It employs two alloy-steel briquetting rolls of 20½ in. diameter by 9¾-in. face and produces almond-shaped briquettes of a size 1½₁₆ × 1¼ × 1 in. thick. To obtain the required briquetting pressures, a load of about 100 tons is applied to each roll by means of springs. The spring assembly is held by two rods extending through the frame, and the spring tension is controlled by adjusting two nuts that hold the spring assembly in place. The rolls are gear driven at 6 to 9 r.p.m. by a 40- to 50-hp. motor, and a fine oil spray on the rolls facilitates the discharge of the briquettes from the pockets. The briquettes are discharged from the press onto a moving belt, at the end of which there is a screen to remove the fines.

The size of the briquettes produced is of course fixed by the shape and size of the pocket and by the amount the rolls separate during briquetting. Even under maximum spring tension, the fin between the briquettes was approximately ½ in. thick. An attempt to reduce the thickness of the fin by substituting solid blocks for the springs only obtained a partial reduction in the fin thickness. In general, wet briquetting is a simple and less expensive operation with higher capacity than dry briquetting. A stronger briquette is produced, and there is freedom from disintegration during reduction. The outstanding advantage of dry briquetting is a higher briquette density, which permits higher charge weights and greater magnesium yield per retort. This advantage can be intensified by the use of an amount of fluorspar equivalent to 1 to 5 per cent of the weight of the dolomite. Such a briquette after calcination gives a briquette with a density of about 2.5 g. per cc.

In commercial work, the Komarek-Greaves press was highly successful and 73 presses were installed at various Defense Plant Corp. plants. The dolomite and ferrosilicon are both ground to 100 to 200 mesh and mixed in the proportion of 82 per cent dolomite to 18 per cent ferrosilicon. The materials contain a large amount of air, which must be eliminated before briquetting. For this purpose the press has a feeder consisting of right- and left-hand screw elements which press the material toward the center of the feeder where it is engaged by an impeller-type paddle which forces the material to the press rolls. The pressure at which the materials are compressed

is 35,000 to 40,000 tons per sq. in. The rolls are gear driven at 6 to 9 r.p.m., a 75-hp. motor being used to drive the press and feeder. The capacity is approximately 4 tons of briquettes per hour.

Apart from the direct bearing of these experiments and practical results on magnesium metallurgy, those who have worked with certain beryllium processes will find them of interest as bearing on their own problems.

Fuel Briquetting.—One type of press, *viz.*, the open-mold horizontal type, is used chiefly in the braunkohle or lignite industry and, to a small degree, in the peat and

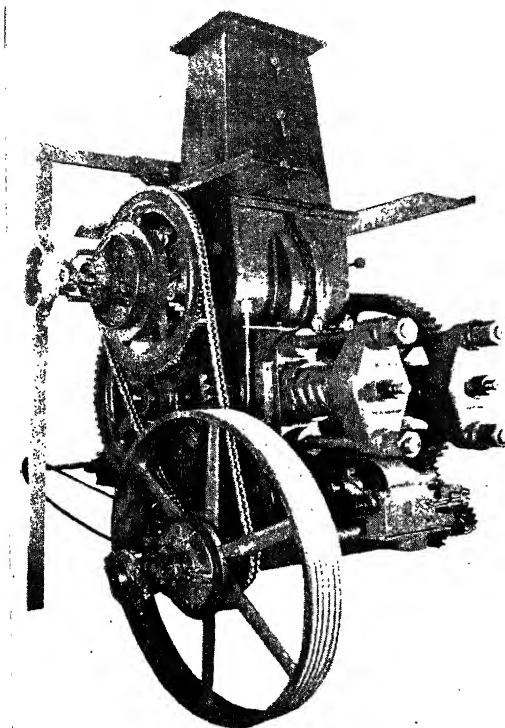


FIG. 8.—Komarek-Greaves press (for normal working pressure of $17\frac{1}{2}$ tons per sq. in.).

sawdust. Wherever this press is used on a commercial scale, no binder is added. Dependence is made upon contained bitumen and the hydrolysis of cellulose. The heated mix, dried to the proper moisture content, is pushed by a piston into a constricted mold, and the constriction of the mold builds up a tremendous pressure on the briquette, together with an enhancement of heat by friction. The row of briquettes already formed—colloquially known as the “rope”—forms the back against which the new briquette is made. The presses are usually run by steam and reciprocate rapidly.

Usually some aftertreatment is necessary before the briquettes are ready for market or for use. Peat blocks or briquettes must be spread in the sun or artificially

dried—usually the former. They dry, shrink, and harden to an excellent marketable fuel. Lignite briquettes, coming from the open-mold press, must be cooled under pressure. This cooling is usually done by continuing their travel in a long channel pushed forward by the press piston. At the end of the channel they are delivered cool, hard, and ready for shipment. Similar treatment has been given to wood waste. Coal or coke briquettes made with oil binders need to be cooled, preferably by travel through the air, or by the less desirable treatment of water sprinkling or immersion before storage—otherwise the contained heat will cause melting and breaking up in the bins. All briquettes made with lime or inorganic binders, whether fuel or metallurgical raw materials, require a hardening treatment, which consists usually of exposure to steam. Practically all inorganic binders require a setting time after the briquettes are made, which may vary from 1 to 24 hr., depending upon the size of the briquette and the character of the materials used. In the briquetting of iron blast-furnace flue dust by the modern corrosion method, an hour's travel is given to the briquettes upon a broad belt, allowing for as great exposure to atmospheric oxygen as possible to permit the formation of a permanent ferric sesquioxide binder. The briquettes leave the press hot and steaming, the heat being caused by chemical action. At the end of the belt travel the briquettes are ready for the furnace, hard and well set. The large briquettes made on piston presses by the same method require a 24-hr. setting time, necessitating a considerable labor cost for handling.

In fuel briquetting a very important phase of the aftertreatment of briquettes consists of some form of carbonization or distillation. The main motive in the employment of such processes rests in the desire to secure a smokeless briquette or anthracite substitute. The recovery of liquid or gaseous by-products is secondary. The carbonization of briquettes made with water-soluble binders—especially sulphite liquor—is gaining ground and has met with a considerable technical success. The expense in this case is comparatively slight, as the briquettes are carbonized in less than half an hour at temperatures between 500 and 700°F. Where tars or pitches have been used as the binding medium, far higher temperatures must be used, and the tendency has been to attempt, unsuccessfully, recoveries to help defray the expense.

In the Carbocoal installation at Clinchfield, Va., briquettes made from semichar were subjected to 1800°F. in closed retorts and the by-product recovered. The binding material used was coal-tar pitch. The briquettes produced were excellent—hard, firm, and smokeless.

In carbonizing briquettes with water-soluble binders, it is the binder that is carbonized, not the coal. It is reduced practically to a charcoal and is, therefore, insoluble in water. The briquettes are carbonized as much for weatherproofing purposes as to gain smokeless combustion.

Binding Materials.—Lime has been used very largely in the briquetting of metal ores and flue dusts. It acts as a mortar either by the formation of calcium carbonate or by the action of free or combined silicate to form calcium silicate. In these cases, treatment in live steam for 8 hr. or setting aside for 24 hr. in the atmosphere is necessary to secure a good setting. The inconvenience of handling such briquettes has resulted in a general abandonment of the use of lime.

Clay.—On account of the long tradition associated with clay as a brickmaking material, the pioneer efforts in briquetting turned to it as the sole admixed binder. In certain sections of Europe the peasants make briquettes from coal dust by mixing with clay and then sun-drying the mix. Where iron ores contain argillaceous material, advantage is taken of its presence in certain instances. Because of its history, the use of clay may be mentioned in discussing briquetting, but it has never proved successful as an admixed binder.

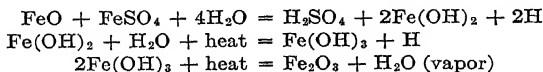
Magnesia.—Good briquettes can be made of inert material having the specific gravity of coal with about 5 per cent oxide of magnesia added with water. The resulting briquettes are tougher than those made from lime, and the set is more rapid. The briquettes, however, are brittle, and the magnesia is expensive. If magnesium chloride is added—making what is known as Sorel's cement—a better bond is secured, but the expense is even greater. Magnesia and magnesia cement are used to assist the bond where they occur in ores and flue dusts, but no commercial plants use them as admixed binders.

Before the First World War, La Société John Cockerill, Seraing, Belgium, was adding magnesium chloride as a binder. As described by Emile Hiertz, the process was as follows: The first press was installed in June, 1910, and the second in March, 1911. They produced 1000 briquettes per hour, weighing 5 kg. each, under a pressure of about 400 kg. per sq. cm. The chloride of magnesium is received from the potash mines at Hanover in tank cars in approximately a 35 per cent solution. The quantity of solution added to the flue dust varies between 2 and 3 per cent, so that the briquettes contain about 1 per cent $MgCl_2$. To fresh dust as much as 10 per cent coke breeze may be added, still making briquettes that have a crushing strength of 25 to 30 kg. after 36 hr. and 45 kg. after 6 days. After treating 40,000 tons of such material, the reports of the blast furnaces show that the briquettes improved the action of a furnace and produced less dust than did Minette ore. As much as 35 per cent of briquettes may be added to the furnace charge without bad effects on operation. It is most interesting to note that the presence of chlorine did not produce any corrosion in the furnace, downtakes, or other piping. The presence of a certain amount of chlorine in the dust reduces the amount of chloride of magnesium which it is necessary to add for briquetting.

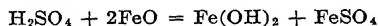
Gypsum.—This, too, has not proved a desirable binder. Briquettes made from it are brittle, and a high percentage is required. Where spent sulphate liquors are available and a limey flue dust is to be briquetted, the two may be mixed, forming an artificial gypsum.

Cement.—*Portland, Hydraulic, and Slag.*—Cement when set is usually some form of calcium silicate. Much has been done in investigating the use of all cements as applied to briquetting. The main objection has been that too long a time is required for the set, as with lime. In Germany the use of slag cement in the briquetting of iron ores and flue dusts was for a time an important development. It was known as the Scoria process and was invented by Prof. Oberschulte and Mathesius. Granulated slag and lime were mixed with ores and flue dusts and exposed to live steam. The cost and the handling and loss of time required have proved handicaps to the introduction of steam hardening in the United States.

Ferric oxide is formed by the corrosion process in flue-dust briquetting by the action of a salt or acid upon iron monoxide, producing chemical reactions that form the ferric hydroxide. The ferric hydroxide dries out, under the heat of the chemical reaction, in a hard mass of sesquioxide of iron. The reagents that may be employed are chlorides, sulphates or sulphites of metals, with the exception of sodium, potassium, and ammonium. Typical reactions are as follows:



As the heat lessens and the bulk of the water passes off, the ferrous sulphate is again formed



Zinc hydroxide and zinc chloride formed in a similar way make an excellent briquetting binder for zinc ores. In the Kippe process a salt is added to zinciferous material to produce a hydroxide reaction. The hydroxide serves as a bond. Copper and other nonferrous flue dusts often contain these reagents and can be briquetted on the roll press with the addition of water only.

Sodium Silicate or Water Glass.—This material makes a good binder for ore briquetting where the metallurgical requirements permit its use. It is, however, expensive and has not received wide recognition in the mineral field. It has been tried often in fuel briquetting. Briquettes are of excellent appearance, but brittle, disintegrated by water, and apt to clinker badly.

Wood Tar.—Wood tar is expensive. It has, in consequence, not been used in metallurgical briquetting. Applied to fuel, it makes excellent briquettes, provided they are carbonized. On account of its cost, however, the only application of wood tars to the briquette industry occurs when charcoal fines occur in quantity in the neighborhood of a wood-distillation plant. The liquid wood tars or pitches can be used for the briquetting of the charcoal, although a large quantity—between 20 and 30 per cent—is needed, depending upon the fineness of the charcoal and the consistency of the binder. Where such large quantities of binder are needed, it becomes advisable to redistill and recover the liquid element from the briquette. Charcoal briquettes so made are denser than the usual product of wood distillation and are excellent for metallurgical purposes. Undoubtedly, the fabrication of charcoal fines into briquettes is an industry that will reach a large tonnage in the future.

Coal and Gas Tar and Pitch.—In the past, the majority of coal-briquetting plants have used heavy-tar or coal-tar pitch binders. The bond is strong and the binder cheap. On the other hand, the smoke is heavy, the odor terrible, and the vapors acid. The use of anthracite briquettes bonded by heavy coal tar was abandoned on an American railroad because the tarry acids generated in combustion were alleged to have caused serious skin eruptions on the train crew. Briquettes made from these binders show considerable degradation in handling and shipping, and in general, it may be predicted that the use of these binders has little future in North America. In Europe, the use of this class of binder for fuel briquetting is practically universal and will continue so. The smoke nuisance is mitigated by the use of piston presses which require a smaller percentage of binder for a good product. Again, the people are accustomed to the burning of soft coal in domestic heating and have little objection to the smoke and odor. In Great Britain it is the practice in most instances to use a pitch of high melting point. The pitch is not melted, but is ground fine and then mixed with the coal, subjected to heat during the mix, and pressed. This method involves larger percentages of binder than does the liquid mix and, while it has advantages of simplicity and low cost of installation, it has made practically no headway in North America. Gas tar and gas-tar pitch are often mentioned as possible briquette binders. In general, they have all the disadvantages of coal tars and pitches to a rather more pronounced degree.

Natural Asphalts.—There is no good reason why natural asphalts should not be a good binder for fuel briquettes, having largely the same characteristics as the oil residuum. The prices of natural asphalts have heretofore been too high to permit much use in briquetting practice.

Asphaltic Petroleum Residuum.—Practically all the asphaltic binders used in commercial briquetting are of this nature, *viz.*, the heavy residues produced by the evaporation and distillation of petroleum. For fuel briquetting, these asphalts have the advantage of being comparatively smokeless; they melt easily and show a high degree of miscibility with coal fines. They are deficient, as compared with coal-tar pitch, in residual carbon, and fuel briquettes made from them have a tendency to

fall apart before combustion is complete. This tendency is usually obviated by adding an amount of binder in excess of the amount required for the bond or by the addition of a small amount of coking soft coal. Up to date, asphalt of the petroleum-residuum variety, having a melting point between 150 and 190°F., has proved the favorite binder for American coals. In fact, the great bulk of coal briquettes today sold on the American market are made with this binder melted to liquidity and mixed with the hot dry coal through well-designed proportioning devices. Some attempts have been made to use such material as binder for metallurgical products and ores, but these have met with little favor. As the briquettes soften in the preliminary heat and at comparatively low temperatures, they are unable to stand the heavy burden—a feature that prevents their use in most metallurgical operations. Some attempts have been made to use water-soluble asphalt binders, usually by treating the asphalt with strong alkali solution and depending on a subsequent baking operation to fix the binder.

Glue.—The various animal glues are too expensive on the one hand and too odorous on the other to be used extensively in any kind of briquetting. In some parts of Europe where hand briquetting is done in homemade molds, some kinds of glue have served as binders.

Starch.—The various starches, having the property of forming a paste when heated with water, make good binders where waterproof briquettes are not required. The preparation of the starch binder with water on a large scale is somewhat difficult. Other binders, such as sulphite liquor, serve equally well, without this added trouble. There have been no briquetting plants operated successfully with starch binder alone. On the other hand, the waterproofing of starch binder with oil has been very carefully studied, and the C. E. Hite binder, used in the operation of the fuel-briquetting plant at Lykens, Pa., is made by the following formula: 25 gal. of water are mixed with 140 lb. of starch, and the mixture is dropped slowly into 175 gal. of boiling water. As the starch paste is formed, 35 gal. of melted asphalt are added very slowly at a temperature of 220°F. When emulsification is complete, the binder is ready. After the briquettes are made, they are dried at a temperature below 400°F., sufficient to dextrinize the starch. This method, involving as it does considerable pains in the binder manufacture, is hardly adapted to the briquetting of ores or flue dusts. There is no question that the anthracite fuel prepared by the above formula is excellent.

Gluten.—If a binder is ever required for the Northwest lignites, gluten, prepared from the Northwest wheat waste, will probably be a factor in the binder market. At present it is not especially prominent.

Molasses.—This forms a good binder, as it has a large proportion of cohesive pectin. Briquettes made with molasses must be made waterproof, preferably by heat. It is today merely a question of price between molasses and other cellulose binders, such as sulphite liquor.

Sulphite Liquors.—Sulphite liquor, a by-product of the sodium sulphite process of papermaking, is produced in enormous quantity. In most places today it is a stream polluter and a nuisance. Evaporated to 50 per cent solids, it is an excellent briquette binder. It has also been used in the making of foundry sand cores. It has an appreciable amount of pectin or pectin equivalent. In the manufacture of fuel briquettes it requires carbonization, whereby a strong odorless and smokeless waterproof briquette is produced. It is not advisable to waterproof this binder by treatment with oil or chemical reagents, even if such procedure is possible. In this case the bad combustion odors still remain. On the other hand, the carbonization of sulphite liquor produces a very strong charcoal, particularly if mastication has been employed in the preparatory treatment. In metallurgical briquetting, and particularly in the handling of blast-furnace flue dusts, it has the advantage of being a duplex binder if in acid condition. On the one hand, the pectin content makes a

briquette strong enough to drop from a roll press (it is essential that the roll press in this instance be properly designed for the purpose), while the acid or salt content acting upon the iron monoxide produces a sesquioxide binder as described under the head of ferric oxide. Briquettes so made season in an hour and are ready for the furnace. As a consequence, sulphite liquor has been largely adopted in the practice of the corrosion process of flue-dust briquetting. It is understood to be the reagent depended upon for briquetting the charge in the continuous distillation process of the New Jersey Zinc Co., in which the entire charge is briquetted.

For such special purposes as this, briquetting will continue to have a place in metallurgy, but there is no question that for briquetting ore and flue dust the rise of the reverberatory furnace and the extension of the field of the Dwight-Lloyd roaster have greatly diminished the importance of briquetting.

Exothermic Briquettes.—An interesting development in briquetting has taken place in the ferrous field, which seems to invite note in nonferrous metallurgy. This is the use of exothermic briquettes to add alloying elements to steel. They produce a turbulence that greatly aids in distributing the added metal thoroughly in the steel. The source of energy is a reaction between sodium nitrate and ferrosilicon, the briquettes being composed of finely ground ferroalloy mixed with dry sodium nitrate and ferrosilicon in such a way that the sodium nitrate and the ferrosilicon are in a more intimate mixture than the ferroalloy and the sodium nitrate. The mixture is briquetted, dried and heated to the melting point of the sodium nitrate, so that this material also acts as a bond to hold the briquettes together. When the briquette is added to the metal bath, a reaction is started between the sodium nitrate and the ferrosilicon, the amounts of these materials being determined by the heat that it is desired to liberate.¹

¹ *Mining Met.*, February, 1944, p. 77.

CHAPTER XIV

POWER PLANT AND ACCESSORIES

BY RODERICK D. DONALDSON¹

Introductory.—Although a power plant is a power plant whether utilized in a metallurgical process or for any other application of power, it should be noted that circumstances permit a design for most metallurgical purposes to secure a greater over-all thermal efficiency than in the usual power plant of like size. The reason for this is that in many metallurgical plants power units can be selected to furnish just the required power for the processing performed. However, it is recognized that for metallurgical purposes the geography and the terrain of the metallurgical power user sometimes dictate the type of power plant to be constructed so that the refinements of power-plant efficiency must be neglected.

Prime Movers and Their Choice.—Prime movers for a metallurgical power plant will be selected first on account of local conditions and second on account of the load characteristic to be served. As many metallurgical plants are located in areas having a poor and scarce water supply, it may be found that internal-combustion engines will best serve the conditions confronting the engineer. Where the water supply is poor and scarce, difficulties may be confronted with steam-plant operation through creation of boiler scale (or the necessity of expensive water treatment) and through inadequate water for condensing purposes.

If internal-combustion engines are selected because of poor and scarce water supply or because of other reasons, elevation of the power plant site above sea level must be given careful consideration as such engines lose their horsepower rating as the elevation increases. Furthermore, the jacket cooling water system should be of the closed-circuit type so that the water once treated is used over and over again with very little make-up. In this system the hot water circulated through the engine jackets flows through pipes over which water drips. This closed system will prevent scale-forming troubles in the water passages around the cylinders and through the engine parts.

If the conditions confronting the engineer dictate the use of internal-combustion engines, the inability of this type of engine to carry overloads must be taken into careful consideration. If the load to be carried by the power plant includes overcoming the inertia of a heavy piece of machinery, special attention must be given to the size of the engines selected and the design of the flywheels on these units. Unless this matter is given careful consideration, the starting of a large crusher or a large rotary kiln may cause a shutdown of an internal-combustion engine plant which is amply large to carry the load when the machinery is in full operation. Should the inertia load of a particular machine be excessive, additional relief can be secured in the case of electric motor-driven machinery by the selection of a motor having more than usual "slip."

Under these circumstances, a synchronous motor should never be used on a machine requiring a high starting torque. It will probably be found that better over-all results can be secured by selecting a motor with more than usual end-ring resistance, thereby sacrificing motor operating efficiency to a greater starting torque. In extreme cases and in large motor application, it may be necessary to go to a specially designed

¹ Consulting engineer, Cresco, Pa.

motor of the wound-rotor type with additional resistance inserted in the circuit which can be cut out when the machine is operating at rated speed, thereby cutting down the rush of current from the power plant. As internal-combustion engines have poor economy when operating at partial loads, it may be well to install one or more small units for utility service such as lights and small power.

Some metallurgical situations may present an opportunity for the development of hydroelectric power. Where such opportunities exist, careful analysis should be made to determine the minimum stream flow either as naturally exists in nature or as it may be augmented by upstream storages. In a great many cases, the minimum stream flow either in the summer drought period or in northern latitudes through winter freeze-up will be found to be in excess of that experienced after the hydroelectric plant is built. When considering the installation of a hydroelectric plant, careful analysis should be made to determine if the investment in the hydroelectric unit can be surely amortized through the profitable life of the metallurgical development. In some cases, it will be found that the fixed charges on the hydroelectric development (including amortization) will be greater than the operating expenses of some other type of power source.

Waste-heat Boilers.—As waste-heat boilers are frequently an efficient and profitable adjunct of a metallurgical plant, the selection of steam-power units may be desirable. Unless there is a profitable use for low pressure, exhaust-steam turboelectric units making use of relatively high pressure from the waste-heat boilers and operating at a vacuum of 2 in. absolute pressure or better should be the selection. While the jet or barometric condenser may represent a smaller initial investment, it is believed that the surface type of condenser will prove more profitable. This is particularly so if the boiler feed water runs above a few grains hardness. It is surely not profitable to waste the condensate from the turbine and be required to treat all the boiler feed water.

Waste-heat boilers are rarely worth installing unless the average temperature of the gases is 1200°F. or more. If it is 2500°F., the boilers may be of standard proportions, except for the grate and furnace. With lower temperature, more heating surface must be used per horsepower than is normal, and the gas velocities must be

TABLE 1. AVERAGE WORKING TEMPERATURES OF WASTE GASES
(Babcock & Wilcox Co.)

Process	Temperature Fahrenheit
Brick kilns.....	2000-2300
Zinc furnaces.....	2000-2300
Copper-matte reverberatories.....	2000-2200
Beehive coke ovens.....	1800-2000
Cement kilns.....	1200-1600
Nickel-refining furnaces.....	1500-1750
Open-hearth steel furnaces.....	1100-1400

increased. This last requirement implies high (usually mechanical) draft. The draft at the primary furnace must not be impaired. The evaporation per square foot of heating surface may fall to 2 lb. or so. The efficiency may exceed that of a boiler burning primary fuel, as there are no combustion losses to be considered. Dust and tar deposits may give trouble, and clean-out doors should be liberally provided. Settling chambers are often necessary. Gases containing sulphur are corrosive in the presence of moisture. Auxiliary grates, burning primary fuel, are sometimes used to steady conditions. If W lb. of gas are available per hour at $T^\circ\text{F.}$, and if the

amount of heating surface is sufficient to reduce the gas temperature to $t^{\circ}\text{F.}$, the horsepower derived is $W(T - t)s \div 33,479$ where s = mean specific heat of gases.

In the ordinary copper-refining furnace about 6 or 7 lb. of water will be evaporated in the waste-heat boiler per pound of fuel fired in the furnace. For a general discussion of the waste-heat boiler, see a paper by Arthur D. Pratt.¹

Even where hydroelectric power is not available, it is probably best to generate electricity and distribute power for mills in that way, rather than by the involved systems of shafts and belting that a steam-driven plant requires.

Choice of Prime Movers.—It is assumed in the selection of power units for a metallurgical plant that irrespective of the type of prime mover selected the power plant will consist of electric generators generating electricity to propel electric motors driving individual machines or driving a short jackshaft for the operation of several small machines. It is difficult to imagine a situation where either operating economies or lower initial investment will be secured through direct engine drive on the machines individually or on a system of shafts and belting.

The problem of fuel transportation to isolated metallurgical plants may be so acute as to render it advisable to have the internal-combustion engine generator or steam electric-generator plant located some distance from the main operations and to transmit the electric power to the main operations.

If the "inexhaustible wood supply" in the area indicates the use of wood for fuel great care must be used to determine the so-called "inexhaustible supply" and determine the cost of logging operations to bring this fuel to the wood-burning plant after the wood in the immediate neighborhood has been used. If wood is being used as fuel, a Dutch oven will be required as an adjunct to the usual boiler furnace, as such fuel requires a much greater cubic space per ton fired than does coal. A wood "hog" or masticator will probably be found necessary so that the wood may be introduced into the Dutch oven in sufficiently small pieces to result in reasonable boiler efficiency.

When selecting engine units, it should be remembered that high-speed units will cost less and weigh less per horsepower developed but that such high-speed units, particularly in the case of internal-combustion engines, carry with them a much more rapid depreciation and a much higher maintenance cost.

Where steam is needed in metallurgical operations in order to heat solutions, a very inefficient noncondensing reciprocating engine may be the proper selection, the engine in this case merely acting as a reducing value with a power byproduct. However, such would be the case only where all the exhaust from such an engine is profitably used in the metallurgical process. Where only very hard water is available with all its attendant boiler troubles, it may be found that a turbogenerator condensing system for the generator of power is preferable. In such case the heat requirements in the plant process may be better secured through the use of a closed heating system rather than permitting the condensate to be lost in the heating process.

The gas engine and the diesel engine are both inflexible units. The amount of air per cycle limits the fuel that can be burned. They do not respond to overloads, and at underloads the efficiency rapidly falls off.

As George J. Young points out,² the wide adoption of the diesel engine in marine service shows its reliability, but in mining districts skilled diesel operators are comparatively scarce as compared with steam engineers.

In wild countries simplicity and reliability are so important that operating costs may be sacrificed somewhat to them. That is, the unreliability of the skilled labor

¹ *Trans. A.S.M.E.*, Vol. 38, p. 599.

² *Eng. Mining Jour.*, June 28, 1919.

supply and the difficulty of obtaining repair parts for complicated apparatus must always be kept in mind.

In engines, generators, and motors, it is usually a good plan to figure on the next larger size than the one recommended by the manufacturers.

An internal-combustion unit should be arranged to start under practically no load, the load to be thrown on gradually after the engine is in operation.

Pipes and Piping.—The caution as to allowing expansion joints in long steam pipes should be unnecessary, as should also be the advice to cover steam pipes, but neglect of these injunctions can be seen in nearly any mining camp.

Bleeder valves should be provided at any low points on either steam or air piping.

Pumps.—The liquids to be moved in metallurgical establishments vary from water to thick pulp and include highly corrosive acids, alkalis, and salts. The pressures likewise vary from a small head encountered in keeping sumps clean, to the high back pressures of filtering operations.

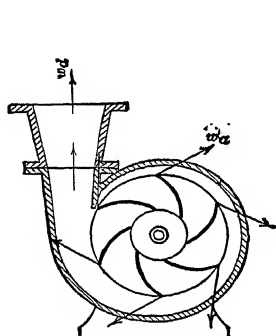


FIG. 1.—Volute pump.

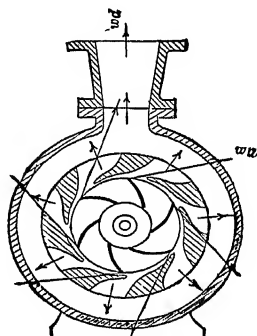


FIG. 2.—Turbine pump.

Reciprocating Pumps.—These may be either of the piston or the plunger type. These types are too well known to need extended description. Plunger pumps are usually used with large quantities of water, which may be dirty, and for high lifts. The plunger pump has less friction and more accessible packing than a piston pump, while the latter will require less space for a given capacity. For handling muddy water, ball valves are probably best.

A reciprocating pump is essentially a slow-speed machine because of the reversal of travel against a practically incompressible medium. If it is desired to keep up a uniform flow and delivery, a duplex or multiplex arrangement must be adopted, and with a single pump large air chambers should be provided to take up the shocks.

A plunger pump should be used for clear solutions only, as grit and dirt rapidly destroy the fit.

A **centrifugal pump** consists of a casing in which revolves an impeller, *i.e.*, a wheel carrying a number of suitably shaped vanes. The centrifugal force sets up a static head in the casing, and a certain velocity is imparted to the liquid. With the discharge valve closed, only the static head is noticeable and can be measured. This is called the shut-off head and is $H_s = u_a^2/2g$ nearly, where u_a = the peripheral velocity of the impeller in feet per second and $g = 32.2$ ft. per sec. When the discharge valve is opened, other forces come into action and it is possible to obtain a head higher

than that due to the centrifugal force alone. This higher head is chiefly obtained by the conversion of the velocity into static head.

This conversion can be accomplished in two ways, resulting in the two types:

1. *Volute Pump*.—The impeller is surrounded by a spiral casing (the volute) of gradually increasing cross section (Fig. 1).

2. *Turbine Pump*.—The impeller is surrounded by a guide ring, provided with a number of suitably shaped channels which receive the liquid in the direction in which it leaves the impeller, guide it to the outside, and at the same time reduce its velocity to such an extent that it will readily turn in any direction and flow toward the point of the casing where the discharge may be located (Fig. 2).

The volute pump is recognized by its spiral casing and tangential discharge nozzle, the turbine by its concentric casing and radial discharge nozzle. The practical limiting velocity and head in a volute pump with one impeller are $U_a = 120$ ft. per sec., with a corresponding head of about 220 ft., while with a well-designed turbine pump U_a may equal 140 ft. per sec., with a corresponding head of 300 ft.

Multistage centrifugal pumps are used where higher heads must be overcome, though this would be most unusual in metallurgical practice.

PISTONLESS PUMPS

Pulsometers (Fig. 3) are pistonless steam-operated displacement pumps with two chambers *AA*. The steam enters these chambers alternately and is controlled by a ball or flap valve *C*. It enters one chamber and forces the water contained therein out through a check valve *F* to the discharge pipe *H* until the steam enters the discharge pipe, where it is condensed and produces a suction effect that throws the valve *C* and draws up the water through the suction valves. *J* is an air chamber. Notwithstanding their low efficiency, they are used extensively in mines, at railroad watering stations, for draining building pits, etc. Suction lifts runs up to 26 ft., maximum, preferably 7 to 14 ft.; discharge head, up to 150 ft. The steam pressure must be about 50 per cent higher than the total water pressure. Low heads give a lower efficiency than high heads. The discharge water is heated 3.5°F. for a head of 30 ft. For every additional 30 ft., add 1.5°. For water temperatures in excess of 120°F., no suction lift is obtainable. The steam consumption by volume is two to three times the water displacement. Wood finds duties varying from 10,500,000 to 13,400,000 ft.-lb., the higher figure corresponding to the higher head. This corresponds to a steam consumption of 190 to 150 lb. per hp. per hr. A duty of 21,345,000 ft.-lb. has been found for total lifts of 102.6 ft.

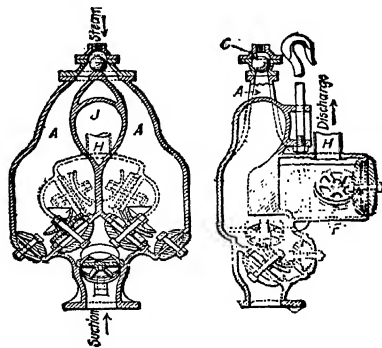


FIG. 3.—Pulsometer.

An ingenious diaphragm pump is made by T. Shriver & Co., Harrison, N.J., for handling gritty materials in filter-press work, the diaphragm being moved by a series of concentric rings on one side and by a crescent-shaped disk on the other.

Compressed-air pumping may be accomplished by displacement apparatus (eggs, montejus, etc.). The air pressure is applied directly on the liquid in the tank. Two displacement tanks are generally used, alternately filling and discharging. The valve distributing the compressed air may be operated by hand, by floats, by the drop in air pressure upon discharge of liquid contents, by the air following the liquid

through the line, or by some suitable timed mechanism. If sufficient submergence can be had, the tanks are submerged and the liquid will then flow in by gravity, as in the Wheeler system. If not, the empty tank may be connected to the suction side of the air compressor and a partial vacuum produced, as in the Harris system. The

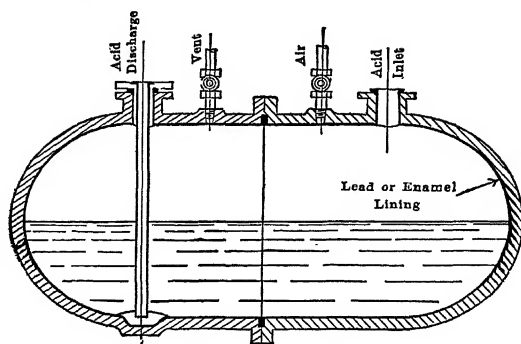


FIG. 4.—Acid egg.

acid egg is of particular advantage in the cheap intermittent handling of highly corrosive fluids.¹

The Starrett air-lift pump² utilizes two displacement chambers from which the air alternately displaces the water by pressure, after which the same air is allowed to escape up the standpipe and, in the manner of the air lift, assists in raising the water

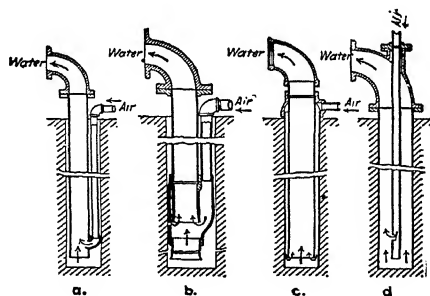


FIG. 5.—Arrangements of air lifts.

and increasing the effective head. A test of such a pump by H. C. Behr³ gave an efficiency of 50 per cent at the compressor, and 35 per cent net.

The air lift consists of a drop pipe placed in a well with its lower end submerged, so that it is under the free pressure of the water, more or less filling the well. An air pipe delivers air at the bottom of the drop pipe and forms a mixture of air and water, which is lighter than the column of solid water in the well; consequently, the mixture rises above the level of the surrounding water. The displacement of the

¹ This apparatus is a container of oval shape similar to Fig. 4. Such an egg-shaped vessel is free from pockets and is readily drained.

² *Eng. Mining Jour.*, Mar. 30, 1907.

³ *Calif. Mining Bur.*, Bull. 9.

air compressor required per gallon of water pumped is $D = kl/585 \log [1 + (s/34)]$ cu. ft., where l = lift in feet, measured from the lower water level, when pumping, to the point of discharge; s = submergence below the lower water level in feet, when pumping; k = a coefficient taking care of all the losses, approximately $k = 3 + (l/700)$. The operating pressure = s -ft. water column is not always known in advance, as the water will stand at a lower level when pumping than when not pumping, and it is the former that determines the point from which the submergence is measured. The central air-pipe system, although obstructing the passage to some extent, has the advantage over the other systems that the submergence can be changed conveniently and thus the point of highest economy determined. The starting pressure = S_a ft., where S_a = submergence when not pumping, is generally higher than the operating

TABLE 2.—ECONOMICAL CAPACITIES AND DIMENSIONS OF AIR LIFTS WITH CENTRAL AIR PIPE

Well, inches	Diameter of		Gallons per minute based on a velocity of 4 ft. per second
	Drop pipe, inches	Air pipe, inches	
4	3	1	70
5	3½	1¼	95
6	4	1¼	125
6	4½	1½	158
7	5	2	195
8	6	2½	280
9	7	2½	380
10	8	3	500
12	10	4	780

TABLE 3.—AIR-LIFT DATA¹

Lift, feet, l	Water horsepower, W	Displacement of air compressor, cubic feet per minute, D	Operating air pressure, pounds per square inch	Air, horsepower, adiabatic, A	Indicated steam horsepower, I	Overall efficiency, per cent, $100 \frac{W}{I}$
20	0.00505	0.38	13	0.017	0.0213	23.7
40	0.0101	0.475	26	0.036	0.045	22.5
80	0.0202	0.68	52	0.0825	0.103	19.6
120	0.0303	0.82	78	0.129	0.162	18.8
160	0.0404	0.97	104	0.178	0.223	18.2
200	0.0505	1.13	130	0.235	0.294	17.2
250	0.063	1.44	162	0.335	0.418	15.0

¹ Based on a submergence of 1.5*l*, this being considered the most desirable submergence. Figures in table are for a capacity of 1 gal. per min.

pressure. In Fig. 5, *a* = Pohlé or side inlet, *b* = annular space inlet, *c* = Saunders system, *d* = central air-pipe system.

The efficiency of the air lift is very low, but its many advantages, *viz.*, simplicity, absence of working parts in the well, ability to handle gritty water or oil, and principally the fact that the full area of the well may be rendered available, tend to counteract its low duty. Generally it is desirable to work a well to the limit of its capacity, regardless of expense.

The best efficiency is obtained for a capacity of 10 to 15 gal. per min. per sq. in. of drop pipe (allowing for an obstruction of 20 per cent by the central air pipe, this will be equivalent to a water velocity of 3.2 to 4.8 ft. per sec.).



Fig. 6.—Bronze clapper valve.

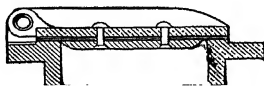


Fig. 7.—Leather-faced clapper valve.

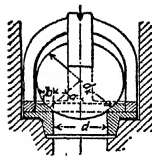


Fig. 8.—Ball valve.

Another method of air pumping is to admit compressed air to the steam end of a steam pump. This can be done without difficulty in a simple-cylinder direct-acting pump, but where the air is used expansively the moisture in the air is apt to freeze at the exhaust nozzle and choke the exit. A remedy is to heat the air before admission; this also effects a considerable increase in efficiency, which at best is quite low.

Air Chambers.—The object of an air chamber is to provide an elastic element in the pipe line and produce a uniform flow. Simple pumps require large air chambers (eight times the displacement of the plunger per stroke); duplex pumps only small air chambers (four times the displacement), duplex low-service (75-lb. pressure) and small general-service pumps require no air chamber. Air chambers are a necessity on all crank and flywheel power pumps. If the air coming in with the water is not sufficient to replenish that absorbed by the discharge water, an air-charging device should be used. This may be an independent air compressor or simply a pipe connected to one of the pump chambers, provided with two check valves for air inlet and outlet. On pumps for over 300-lb. pressure, where the air is absorbed rapidly by the water, a spring-loaded ram, called an alleviator, should be used.

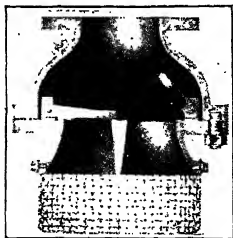


Fig. 9.—Clack valve and strainer.

A vacuum chamber performs the same duty on the suction pipe and should have its inlet on the dead end of a tee so as to be in line with the current. It should be placed on the suction pipe of every good-sized pump and is a necessity on high lifts and pumps producing an irregular flow. When the water carries much air, the vacuum chamber may act as an air collector from which the air may be removed by a vacuum pump. In long suction pipes where the air has more time to become liberated from the water, the chamber may be constructed as a separating box.

The object of a foot valve is to keep the pump primed during a stoppage. It adds to the friction and reduces the possible suction lift. Figure 9 shows a clack valve and strainer. If the water carries impurities that might do damage to the pump, a strainer should be provided in the suction pipe.

The ideal suction lift of a pump equals the water column sustained by the atmospheric pressure minus the vapor pressure; the former is affected by the barometer and altitude, the latter by the temperature of the water. The hydraulic losses, which comprise the friction through the suction pipe, valves, and passages, and the velocity head are generally insignificant as compared with the influence of the air contained in the suction water, the effect of which cannot be computed.

Acid Pumps.—When an acid or other corrosive substance is to be handled by a pump, no material should be used that is attacked by the substance. Therefore, pumps may be constructed either wholly of brass, bronze, rubber, or even stoneware, or wholly of iron, or they may be merely bronze fitted, which means that certain working parts, such as pistons, rods, and pump pistons, are made of bronze or are bronze lined. In ordering bronze or brass-fitted pumps, the parts that should be made of bronze or brass should be clearly specified. In an all-iron pump the pump cylinder is bored to size and has no lining. The valve seats and stems are made of malleable iron.

If, in a brass-fitted pump, the size permits, the piston rod should be split in the crosshead, and the steam piston rod should be made of steel. Brass is too soft to wear well in the high temperature maintained in the steam cylinder.

Rotary Pumps.—In rotary pumps not of the centrifugal type a pair of spur gears run in a closed casing. The points of the teeth, which run close to the case, pick up the liquid and hold it in the series of pockets formed by the teeth, then discharge the liquid on the delivery side. The shafts are driven by exterior gearing, which should be on both sides of the casing, so that the pump will not itself act as a gear.

Solutions can be delivered by these pumps against a high head, but their mechanical efficiency is low, and the liquid to be pumped must be clear.

The **Frier spiral sand pump** consists of a rotating helix which scoops up the material to be handled, passes it through the spiral passage, and discharges it at the center into a lift pipe.

The principle appears also in the device for the continuous feeding of tube mills and is the basis of the spiral wheel for pumping molten metal out of lead kettles.

The **ejector or steam siphon** is often used for moving liquids where an admixture of water will do no harm. The principle is familiar in the injector for steam boilers.

The principle is used with molten metals, to produce finely divided powders by steam suction and "atomization."

Operating Direct-acting Pumps.—Small pumps are provided with a petcock on the discharge chamber for blowing out the air. On larger pumps a starting valve, or waste valve, should be placed on the discharge pipe inside of the gate and check valves. Prime and start the pump with the waste valve open to work the air out of the pump chambers. If there is a check valve, this will open automatically when the right speed is reached, but if there is only a gate valve a gauge should be placed at some point inside of the gate valve, which will indicate the proper time to open the gate valve, *viz.*, when the pressures on both sides are balanced. Both sides of a duplex pump should be packed alike, or one side will travel at a higher velocity than the other.

Before opening the throttle valve, blow out the steam pipe. If cylinders are jacketed, admit steam to the jackets some time before starting. If the air pump is independent, start it and get up a vacuum before starting the main engine. After stopping, open all blowoffs and drain the valves. If the pump is to be idle for some time, open the cock of the oil cup, so as to let the oil flow into the steam chest, then let the pump make a few strokes to distribute the oil well over the inside of the steam end. When a pump is to be drowned, flood the cylinders with oil and shut off steam from the surface.

Keep the stuffing boxes well packed with a good quality of packing. Screw up the gland no tighter than is necessary to stop leakage, and renew the packing before it becomes hard. Watch the rods and the outside-packed plungers to detect cutting. If the pump is provided with a surface condenser, never exhaust into an empty condenser, as this will destroy the tube packing.

Operating Centrifugal Pumps.—Centrifugal pumps are primed before starting, either by an ejector with the discharge closed, or, if a foot valve is provided at the end of the suction pipe, it may be filled with water through a priming pipe, the air escaping through a vent at the highest point of the casing. Start and bring up to speed, then open the discharge valve. Always close the discharge valve before starting and stopping.

The stuffing boxes should be packed with square graphite packing. Round packing will wear grooves in the shaft. Obtain from the manufacturer a set of characteristic curves and compare with actual performance. If the readings do not agree with the curves, examine the pump for defects, such as stones in the impeller or broken blades. These curves will also assist in running the pump near the point of maximum efficiency.

Setting up Pumps.—The suction pipe should be large and as short and direct as possible. Centrifugal pumps will not operate satisfactorily on suction lifts of over 6 to 10 ft. on long suction lines. Bends should be of long radius. Lay to a uniform grade, rising toward the pump, so as to avoid air pockets. In connecting the steam pipe, make due allowance for expansion. Place a throttle valve in the pipe close to the pump, and provide means for draining the pipe with the valve closed. Before connecting the pipe to the steam chest, blow out the steam thoroughly to keep dirt from getting into the cylinder.

The nozzles of a centrifugal pump as furnished by the manufacturer are necessarily computed for fairly high velocities. Increases should be attached, and the pipes should be made large enough to give velocities of 4 to 5 ft. per sec. in the discharge pipe and 3 to 4 ft. per sec. in the suction pipe. A strainer of liberal proportions should be at the bottom of the suction pipe, and the free opening through the strainer should be two to four times that of the pipe. The openings through the strainer should be smaller than the smallest passage in the impeller. Install a gate valve and a check valve near the discharge nozzle with suitable drains, priming pipes, and a pressure gauge.

GAS CONVEYING

The movement of gases is of prime importance in metallurgy, for the basic operations of fusion, oxidation, and reduction are dependent on air or gas supply. This supply may be governed by natural draft (chimneys) or by fans, compressors, or blowers.

CHIMNEYS

Chimneys.—Apart from the function of producing draft, metallurgical stacks are called upon to carry off noxious fumes, and must be built to withstand their effects. In addition, they must be designed to withstand wind stresses, often far in excess of the stress placed upon the stack by its own weight.

Sulphur dioxide will not harm either brick or steel stacks so long as the temperature is at all elevated. Sulphur trioxide becomes harmful at temperatures under about 400°F., if there is any great concentration of sulphur trioxide and water vapor. Above this point the trioxide has little effect on an acidproof mortar. Chlorine and nitrous acid fumes are about the same—an impervious brick should be used, with

waterproof mortar, and the temperature must be kept fairly high. Sulphur trioxide is practically quantitatively removed by the Cottrell process.

If the temperature of the gas is low and the sulphuric acid is not removed by a Cottrell apparatus, projecting lead petticoats dripping into a pan in the bottom will do some good, though they check the draft.

Where lightning rods are installed on acid chimneys, the upper 50 ft. or more of the complete rod should be sheathed to protect the copper from the effects of the acid. Lead covering is in most cases effective. All chimneys handling acid gases should be equipped with an outside ladder, the upper portion of which should be covered with lead or an acid-resisting material. Chimneys that have been in practically continuous service for years without showing any effect from the smoke stream have been observed to develop defects, particularly in the upper portions, after they have been shut down for a protracted period.

Although the conditions of temperature, dilution, acid mixture, and the like may be such as not to cause damage while the chimney is in operation, yet an accumulation of dust on the inner walls, which is deliquescent by virtue of its acid content, may tend to do damage when the chimney is not in operation.

The weather, rain, fog, snow, or a heavy humid atmosphere furnishes the necessary water within the chimney to convert the previously inert dust with an acid content into a liquid acid which immediately becomes active.

It is, therefore, wise, when the chimney is shut down for a period, to cover the entire opening at the top with a temporary weatherproof lid. This can be made in sections of light wood easily placed and removed. Lugs protected against acid action

TABLE 4.—WEIGHT OF AIR AT VARIOUS TEMPERATURES, PRESSURES, AND DEGREES OF HUMIDITY

Temperature, degrees Fahrenheit	Weight of 1 cu. ft. of dry air (at 14 lb. per square inch or 28.5 in. of Hg), pounds	Increase or decrease of weight for each 0.1 lb. change in pressure, pounds	Increase or decrease of weight for each 1 in. of Hg change of pressure, pounds	Decrease of weight for each 10 per cent increase in relative humidity, pounds
32	0.07688	0.000549	0.002698	0.000019
35	0.07642	0.000546	0.002681	0.000021
40	0.07565	0.000540	0.002654	0.000025
45	0.07490	0.000535	0.002628	0.000030
50	0.07417	0.000530	0.002602	0.000035
55	0.07340	0.000525	0.002580	0.000040
60	0.07272	0.000520	0.002554	0.000051
65	0.07203	0.000515	0.002530	0.000059
70	0.07134	0.000510	0.002506	0.000070
75	0.07068	0.000505	0.002482	0.000081
80	0.07003	0.000500	0.002457	0.000095
85	0.06938	0.000495	0.002432	0.000111
90	0.06875	0.000490	0.002408	0.000127
95	0.06811	0.000485	0.002384	0.000147
100	0.06752	0.000480	0.002359	0.000172
105	0.06694	0.000475	0.002334	0.000199

should be built into the chimney head to which the sections of the temporary lid may be fastened.¹

Theoretically, if H be the height of the chimney, α the expansion coefficient of the gas (approximately 0.00366 per degree centigrade), h the suction head of the chimney, and t' and t the average temperatures inside and outside of the chimney, $H(1 + \alpha t') = (H + h)(1 + \alpha t)$ or $h = \frac{H\alpha(t' - t)}{1 + \alpha t}$. As $1 + \alpha t$ does not differ greatly from unity, it can be neglected, or $h = H\alpha(t' - t)$.

The theoretical velocity with which outside air would enter the chimney (*i.e.*, if there were no chimney leaks or resistance) would be $v = \sqrt{2gh}$ where $g = 32.16$ ft. per sec. Substituting, $v = \sqrt{2gH\alpha(t' - t)}$.

The quantity of air Q passing through a chimney is equal to the area A times the velocity times the weight per cubic foot of the gases (d' at temperature t').

If d_0 is the temperature at 32°F. and 760-mm. pressure and the barometric pressure prevailing is p' ,

the n
$$d' = \frac{d_0}{1 + \alpha t'} \cdot \frac{p'}{760};$$

hence

$$Q = \frac{A \sqrt{2gH\alpha(t' - t)}}{760} \cdot \frac{d_0 p'}{1 + \alpha t'}$$

It should be noted that A in the above expression is not equal to the total real area of the stack, but is a smaller quantity, the effective area. If A_a is the actual area, the effective area will be approximately

$$A = A_a - 0.6 \sqrt{A}$$

While a chimney four times as high will apparently double the draft, actually there comes a point where added friction and cooling by radiation more than overcomes the added draft effect. Chimneys are rarely over 150 ft. high, and when they are it is usually in order to distribute objectionable fumes over a wide area rather than for draft purposes.

Draft Apparatus.—The draft of a chimney varies directly as its height above the grates. The volume of gas it discharges varies directly as the area, and as the square root of the height. Very low and very high chimneys are both undesirable. For the sizes required, see the accompanying tables.

Table 5 gives the horsepower of chimneys computed by Kent method.

For the pounds of coal burned per hour for any given size of chimney, multiply the figures in the table by 5. Chimneys 25 per cent larger are recommended for low-grade bituminous coal, while with oil fuel the rated power will be from three times in the small-size stacks to about two in the large.

The cheapest and least durable chimney is the guyed steel stack. Self-supporting steel stacks and reinforced concrete come next. Radial and hollow-brick stacks, properly designed, are cheaper than those of common brick of equal excellence.

For a chimney 60 ft. high, about 116 lb. of anthracite coal will be burned per hour per square foot of effective area of chimney; 135 lb. for an 80-ft. stack; 152 lb. for a 100-ft. stack; 93 lb. for a 40-ft. stack.

The combustion of 1 lb. for bituminous coal gives about 550 cu. ft. of products of combustion, measured at 500°F. Usually with bituminous coal the stack area is made one-fifth to one-half the area of a furnace grate and one-eighth to one-sixth the area of a boiler grate.

¹ The last three paragraphs are taken from a staff report of the Alphons Custodis Chimney Construction Co.

TABLE 5.—HORSEPOWER OF CHIMNEYS

Diameter, inches	Height								Effective area, square feet	Actual area, square feet
	60 feet	80 feet	100 feet	125 feet	150 feet	200 feet	250 feet	300 feet		
18	25	29	30	36	40	46	51	56	0.97	1.77
24	54	62	69	78	85	98	110	120	2.08	3.14
30	92	107	119	133	146	169	189	206	3.58	4.91
36	141	163	182	204	223	258	288	315	5.47	7.07
42	200	231	258	289	316	365	408	447	7.76	9.62
48	269	311	348	389	426	492	549	602	10.44	12.57
60	437	505	565	632	692	800	894	979	16.98	19.64
72	646	747	835	934	1,023	1,181	1,320	1,447	25.08	28.27
84	896	1,035	1,157	1,294	1,418	1,637	1,830	2,005	34.76	38.48
96	1,186	1,370	1,532	1,713	1,876	2,167	2,423	2,654	46.01	50.27
108	1,517	1,751	1,959	2,054	2,392	2,770	3,098	3,393	58.83	63.62
120	2,180	2,438	2,557	2,986	3,448	3,855	4,223	73.22	78.54
132	2,656	2,970	3,114	3,637	4,200	4,696	5,144	89.18	95.03
144	3,554	3,726	4,352	5,027	5,618	6,155	6,672	106.72	113.10
168	4,878	5,115	5,974	6,899	7,713	8,449	9,146	146.50	153.94
192	6,724	7,852	9,068	10,138	11,105	11,925	192.56	201.06
216	9,987	11,532	12,894	14,123	14,900	244.90	254.47
240	12,378	14,293	15,980	17,505	18,116	303.53	314.16

For measurement of draft pressures, probably the most delicate apparatus is Iles' modification of the Fallis manometer.¹ The Ellison differential draft gauge is also a very sensitive instrument.²

Blast Apparatus.—Forced blast is a necessity for many metallurgical operations, either for ventilation, to secure more rapid combustion than would otherwise take

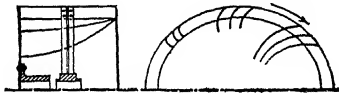


FIG. 10.—Multiblade fan.



FIG. 11.—Paddle-wheel fan with cast casing.

place, or for the maintenance of reducing or sulphatizing atmospheres. The usual pieces of apparatus are injectors and exhausters, disk and centrifugal fans, rotary blowers, and blowing engines.

Fans.—These are used only at low pressures. As said above, they are of two types, the propeller and the centrifugal. The former moves the air in a direction parallel to its axis of rotation, the effect being similar to that of a ship's propeller. The centrifugal fans have an inlet at the center and discharge at the periphery.

Fans may operate with equal efficiency at any speed which their mechanical strength safely allows. They may also discharge, in practice, through orifices varying in area from that of their full discharge, to a very small percentage of the full outlet.

¹ Described in *Eng. Mining Jour.*, Vol. 59, 1900, p. 765; also in Iles' "Lead Smelting."

² American Steam Gauge & Valve Mfg. Co., Boston, Mass.

The efficiency varies with different orifice areas, but not according to any fixed law for all designs.

The horsepower for any fan or blower, if the area of the discharge orifice remains unchanged, varies directly as the cube of the speed; also, if the area of the discharge orifice remains unchanged and the speed is constant, the horsepower varies directly with the density of the gas being handled. The horsepower of a centrifugal fan, if the speed remains unchanged, generally decreases as the area of discharge orifice is decreased. In some designs the horsepower with rated orifice is not over one-third



FIG. 12.—Steel-plate cone fan.

of the horsepower with orifice equal to area of fan outlet, and in other designs it is very little less at rated orifice than with unrestricted orifice.

The horsepower of a propeller fan, if the speed remains unchanged, increases as the area of the discharge orifice is decreased. In some designs the horsepower with rated orifice is not over one-quarter of the horsepower with complete restriction. The propeller fan is used in heating, ventilating, cooling, drying, etc., rarely for pressures exceeding 1 in. of water. Rated capacities are given in Table 6.

TABLE 6.—RATED CAPACITIES OF PROPELLER FANS

Wheel diameter, feet	Free delivery			Static pressure, inches								
				0.25			0.5			1.0		
	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
2	4,000	170	0.24	2,100	710	0.53	3,000	1,000	1.5	4,250	1,420	4.2
3	9,000	465	0.54	4,800	470	1.2	6,750	670	3.4	9,500	940	9.6
4	16,000	350	0.95	8,500	355	2.1	12,000	500	6.0	17,000	710	17.0
6	36,000	233	2.15	19,000	235	5.1	27,000	335	14.5	38,000	470	41.0
8	64,000	175	3.8	34,000	175	8.5	48,000	250	24.0	68,000	355	68.0

Approximate Horsepower of Propeller Fans.—The horsepower required to drive any propeller fan may be represented by an expression of the form

$$\text{Horsepower}$$

where K_1 is a constant depending upon the design and upon other conditions, T is the peripheral velocity or the tip speed of the wheel in feet per minute, A is the gross area of the wheel in square feet, w is the absolute density of the gas handled in pounds per cubic foot, 0.075 is the weight of standard air at 65°F. in pounds per cubic foot.

TABLE 7.—CONSTANTS FOR PROPELLER FANS
(B. F. Sturtevant Co.)

Type of Fan	K_1
Sturtevant propeller.....	0.0006
Multiblade disk.....	0.0012
Blackman propeller.....	0.0012
Davidson propeller.....	0.0015

Centrifugal fans find a much wider application than do the propeller. They are used up to pressures of about 16 oz., or about 27 in. of water.

TABLE 8.—TABLE OF CONSTANTS FOR CENTRIFUGAL FANS

(B. F. Sturtevant Co.)

TYPE OF FAN W = (WIDTH; D = DIAMETER) K

Paddle wheel with 6 to 10 straight blades and $W = 0.45D$...	0.0060
Steel pressure with 24 backward curved blades and $W = 0.15D$	0.0050
Slow speed conveying with 18 forward curved blades and $W = 0.25D$	0.0100
Single-inlet multiblade drum with 60 narrow curved blades, inclined forward and $W = 0.50D$	0.0350
Double-inlet multiblade drum with 60 narrow curved blades, inclined forward and $W = 0.25D$	0.0250
Double-inlet multiblade drum with 60 narrow curved blades, inclined forward and $W = 0.50D$	0.0400
Double-inlet multiblade drum with 60 narrow curved blades, inclined forward and $W = 1.00D$	0.0350

TABLE 9.—CONVERSION TABLES FOR AIR PRESSURES

PRESSURES IN INCHES OF WATER CORRESPONDING TO OUNCES PER SQUARE INCH

Pressure, ounces per square inch	Decimal parts of an ounce									
	0.0	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9
0	0.17	0.35	0.52	0.69	0.87	1.04	1.21	1.38	1.56
1	1.73	1.90	2.08	2.25	2.42	2.60	2.77	2.94	3.11	3.29
2	3.46	3.63	3.81	3.98	4.15	4.33	4.50	4.67	4.84	5.01
3	5.19	5.36	5.54	5.71	5.88	6.06	6.23	6.40	6.57	6.75
4	6.92	7.09	7.27	7.44	7.61	7.79	7.96	8.13	8.30	8.48
5	8.65	8.82	9.00	9.17	9.34	9.52	9.69	9.86	10.03	10.21
6	10.38	10.55	10.73	10.90	11.07	11.26	11.43	11.60	11.77	11.95
7	12.11	12.28	12.46	12.63	12.80	12.97	13.15	13.32	13.49	13.67
8	13.84	14.01	14.19	14.36	14.53	14.71	14.88	15.05	15.22	15.40
9	15.57	15.74	15.92	16.09	16.26	16.45	16.62	16.79	16.96	17.14

PRESSURE IN OUNCES PER SQUARE INCH CORRESPONDING TO INCHES OF WATER

Head, inches	Decimal parts of an inch									
	0.0	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9
0	0.06	0.12	0.17	0.23	0.29	0.35	0.40	0.46	0.52
1	0.58	0.63	0.69	0.75	0.81	0.87	0.93	0.98	1.04	1.09
2	1.16	1.21	1.27	1.33	1.39	1.44	1.50	1.56	1.62	1.67
3	1.73	1.79	1.85	1.91	1.96	2.02	2.08	2.14	2.19	2.25
4	2.31	2.37	2.42	2.48	2.54	2.60	2.66	2.72	2.77	2.83
5	2.89	2.94	3.00	3.06	3.12	3.18	3.24	3.29	3.35	3.41
6	3.47	3.52	3.58	3.64	3.70	3.75	3.81	3.87	3.92	3.98
7	4.04	4.10	4.16	4.22	4.28	4.33	4.39	4.45	4.50	4.56
8	4.62	4.67	4.73	4.79	4.85	4.91	4.97	5.03	5.08	5.14
9	5.20	5.26	5.31	5.37	5.42	5.48	5.54	5.60	5.66	5.72

Multiblade fans are made by the American Blower Co., Sirocco; the Buffalo Forge Co., Conoidal; Keith & Blackman, Keith; the N. Y. Blower Co., Seri-Vane; the B. F. Sturtevant Co., Multivane. The wheels contain 30 to 60 blades, generally of shallow radial depth and curved with concave surface forward in rotation. The inlet is exceptionally large compared with the wheel diameter. A spiral casing encloses the wheel (see Fig. 10). The range in pressure is ordinarily 0 to 5 in. of water.

Wheel diameter, feet	Approximate height, feet	Static pressure, inches											
		0.5			1			2			4		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
1	2	1,170	780	0.30	1,650	1,100	0.85	2,340	1,560	2.4	3,300	2,200	6.8
2	4	4,650	390	1.2	6,600	550	3.4	9,300	780	9.6	13,200	1,100	27.0
4	8	18,700	195	4.8	26,400	275	13.6	37,400	390	38.5	52,800	550	109.0
6	12	42,000	130	10.8	59,400	184	30.5	84,000	260	86.0	118,800	368	244.0
8	16	75,000	98	19.1	105,600	138	54.0	150,000	196	153.0	211,200	276	432.0
10	20	117,000	78	30.0	165,000	110	85.0	234,000	156	240.0	330,000	220	680.0

Cast casing paddle-wheel fans are made by the American Blower Co., type V; the Buffalo Forge Co., type B; the Green Fuel Economy Co., cast-iron volume fan; the N. Y. Blower Co., Peerless; the B. F. Sturtevant Co., Monogram.

Wheel diameter, inches	Approximate height, inches	Static pressure, ounces											
		1			2			4			6		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
10	19	360	2,050	0.18	510	2,900	0.50	720	4,100	1.45	880	5,000	2.65
15	27	810	1,390	0.40	1,150	1,970	1.15	1,620	2,780	3.2	1,980	3,400	5.9
23	42	2,150	885	1.05	3,050	1,250	2.9	4,300	1,770	8.4	5,250	2,170	15.5
33	58	4,650	630	2.3	6,450	890	6.5	9,300	1,260	18.5	11,400	1,540	34.0

The wheels are of paddle-wheel type—six to eight blades—and not quite so wide axially as the paddle wheels in “steel-plate” fans. Casings are spiral in form and of cast iron (see Fig. 10). The range of pressures is 0 to 6 oz., and they are used for forges, oil furnaces, light-dust collecting, etc.

Cast-iron and steel-plate pressure fans are made by the American Blower Co., type P; the Buffalo Forge Co., type P; the B. F. Sturtevant Co., “steel pressure blower.” The wheels are of paddle type, with 6 to 24 blades, width narrower than other types. The casing is of spiral form. The range of pressures is 4 to 16 oz., and they are used for forges, oil furnaces, cupolas, gas producers, boosters, etc.

Wheel diameter, feet.	Approximate height, inches	Static pressure, ounces											
		4			8			12			16		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
14	25	600	2,800	1.3	850	3,960	3.7	1,040	4,850	6.7	1,200	5,600	10.4
21	39	1,300	1,900	2.8	1,850	2,680	7.9	2,250	3,300	14.5	2,600	3,800	22.5
34	59	3,400	1,150	7.1	4,800	1,625	20.0	5,900	2,000	37.0	6,800	2,300	57.0
45	77	6,300	875	13.0	8,900	1,240	37.0	10,900	1,500	67.0	12,600	1,750	104.0

Steel-plate-paddle-wheel fans are made by the American Blower Co., the Buffalo Forge Co., the Garden City Fan Co., the Green Fuel Economy Co., the N. Y. Blower Co., the B. F. Sturtevant Co., etc. The wheels have 8 to 12 blades which are deep radially and connected to spider arms emanating from a central hub. Casings are of spiral form. The range of pressures is 0 to 5 in.

Wheel diameter, feet	Approximate height, inches	Static pressure, inches											
		0.5			1			2			4		
		Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
2	40	1,800	510	0.37	2,500	720	1.1	3,600	1,020	3.0	5,000	1,440	8.6
3	60	4,000	340	0.85	5,700	480	2.4	8,000	680	6.8	11,400	960	19.0
4	80	7,200	255	1.5	10,200	360	4.3	14,500	510	12.0	20,400	720	34.0
6	120	16,000	170	3.4	22,800	240	9.6	32,000	340	27.0	45,600	480	77.0
8	160	28,700	128	6.0	40,700	180	17.1	57,500	255	48.0	81,400	360	137.0
12	240	65,000	85	13.5	91,200	120	38.4	130,000	170	108.0	182,000	240	307.0

Steel-plate cone fans are made by the American Blower Co., the Buffalo Forge Co., the Green Fuel Economy Co., the N. Y. Blower Co., the B. F. Sturtevant Co.

Wheel diameter, feet	Static pressure, inches											
	0.25			0.5			1.0			1.5		
	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower	Cubic feet per minute	Revolutions per minute	Brake horsepower
3	4,300	320	0.63	6,100	450	1.3	8,600	640	5.0	10,400	785	9.2
4	7,650	240	1.1	10,800	340	3.2	15,300	480	8.9	18,700	590	16.3
6	17,200	160	2.5	24,300	225	7.1	34,400	320	20.0	42,000	390	37.0
8	30,500	120	4.5	43,000	170	12.7	61,000	240	36.0	75,000	295	66.0
12	69,000	80	10.0	98,000	113	28.0	138,000	160	80.0	169,000	195	147.0

These fans are of the paddle-wheel type, but built on a back or supporting cone instead of a spider having arms emanating from a central hub; 8 to 12 blades are employed, and no casing is used. The fan discharges directly into the atmosphere or into a large room, as the case may be. The pressures run from 0 to $1\frac{1}{2}$ in., and the fans are used for heating, ventilating, cooling, drying, etc.

After a few months' service most fans give materially less than the maker's rating, and allowance should be made for this in buying a fan.

Steam-jet Blowers.—Steam jets have long been used for "blowing" or exhausting in order to maintain combustion in locomotive boilers, usually employing the exhaust from the engines through properly shaped "nozzles." This type of air compressor or exhauster is also extensively used for emergencies and at times in permanent installations for removing foul air from mines, factories, ship holds, and for gas exhausters, for securing forced draft, and for handling gases under low pressures in certain chemical industries. Among its advantages are simplicity, ease of operation, small space, minimum of repairs, and ease of regulation.

A. von Ihering ("Die Gebläse") reports some tests as to steam consumption which indicate 1.3 to 3.4 lb. of steam required per 1000 cu. ft. of air handled, the larger sizes being the more economical.

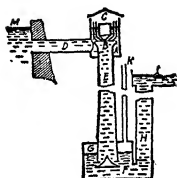


FIG. 13.—Taylor hydraulic air compressor.

Hydraulic Compressors.—Several devices have been made for utilizing falling water for the purpose of compressing air without the use of any mechanical moving parts. The most successful of these is the Taylor compressor, shown diagrammatically in Fig. 13. In the figure, air tubes are represented at *C*, all terminating at the conical entrance *B* to the down-flow pipe *E*. The water supply is furnished through the flume *D*. As the water falls it draws air through the small tubes, carrying it down to the separating tank *G*, where it is liberated at a pressure depending on the weight of water in the vertical pipe *H*. The

compressed air is then conducted through the pipe *K* to the place where it is to be used. The distance from *M* to the tail race *L* represents the height or fall of water that is available.

In this system the compression is isothermal, and the compressed air is saturated with moisture. The oxygen content of the air is reduced, which renders the air less beneficial for mine ventilation if the exhaust from the air tools is planned to assist ventilating. The system offers a very simple solution for utilizing water powers when the market for compressed air justifies its installation. It has the advantage of

Air measurements, absolute pressures				Water measurements			
Free air, cubic feet per minute	Free air, pounds per square inch	Compressed air, pounds per square inch	Horse-power	Cubic feet per minute	Head, feet	Horse-power	Efficiency, per cent
10,580	14	128	1,430	13,057	70.5	1,741	82.17
11,930	14	128	1,623	14,820	70.0	1,961	82.27
9,238	14	128	1,248	12,716	70.6	1,700	73.50

simplicity with a minimum of operating expense, and very high efficiencies are secured. The first cost of the installation is likely to be high.

Figure 14 illustrates some of the dimensions of a Taylor hydraulic compressor installed near Cobalt, Ont. This was designed for a capacity of 40,000 cu. ft. of free air per minute to be compressed to a gauge pressure of 120 lb. The compressed air is conducted to mines through 9 miles of 20-in. pipe leading to two 12-in. lines with a total distributing line of 21 miles in length. The water is admitted through suitable gates to two "heads" each 16 ft. in diameter and containing 66 pipes 14 in. in diameter. The size of the heads is reduced in diameter to about 8 ft., and the whole apparatus can be raised or lowered as required by operating conditions. A cone *a* assists in separating the air and water, and the long horizontal tunnel permits quite complete separation. The compressed air is removed through the pipe *c*, and the water freed from the entrained air escapes through the vertical shaft *b*. Pipe *c* acts as a relief for a surplus of compressed air. Its end is normally below the surface of the water in the tunnel, but if too much air should accumulate the end would be exposed and permit the escape of the surplus without seriously affecting the normal air pressure of the distribution system.

Piston Compressors and Blowers.—The large quantities of air required for blast-furnace and Bessemer converters are usually supplied by piston compressors of large

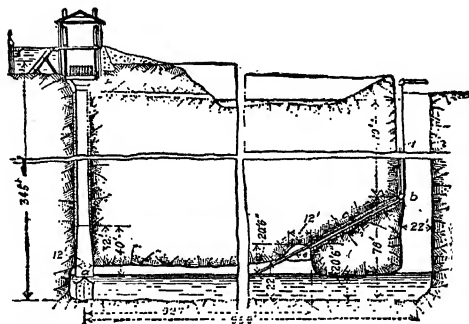


FIG. 14.—Taylor hydraulic compressor installation near Cobalt, Ont.

capacity, driven either by steam or gas engines. Turboblenders directly driven by steam turbines, however, have been developed for this work.

Piston compressors used as blowers for blast furnaces and Bessemer converters usually operate at discharge pressures of 15- or 20-lb. gauge. But small compressors of this type are built for pressures as low as 1 lb. above atmosphere, single-stage up to 80 to 100 lb., two-stage from 80 to 500 lb., three-stage to about 1200 lb. (small units even to 2500 lb.), and four-stage to 3000 lb. or even higher. The tendency in blower design is to secure increased capacity by higher speeds than formerly. The Allis-Chalmers Co. now uses a maximum speed of 90 r.p.m. for a 48-in. stroke and 85 r.p.m. for a 60-in. stroke, giving piston speeds of 720 to 850 ft. per min. With piston speeds approximating 750 ft. per min. the inlet area is approximately 13 per cent and the outlet area 11 per cent of the piston area. With increased speeds the tendency is to increase clearance, which will usually vary $9\frac{1}{2}$ to $11\frac{1}{2}$ per cent.

The Slick blowing tub consists of a reciprocating cylinder on the outside of the compressing cylinder, arranged so as to open ports at the ends of the cylinders for inlet. This gives unobstructed inlet areas of 18 to 20 per cent with very small clearance.

The ratio of inlet valve area to piston area varies from 0.05 to 0.14. For ordinary types of valves, the inlet area should, as a rule, be not less than 8 or 10 per cent of the piston area. Automatically operated inlet valves are apt to be irregular in their action and reduce the volumetric efficiency of the compressor. This disadvantage is overcome by using mechanically operated inlet valves, usually of a Corliss type, but on account of the variation of discharge pressure they are not used so often for discharge or for inlet on the high-pressure stages of a multistage compressor. The area of the discharge valves will usually vary from 10 to 15 per cent of the piston area, the larger percentage being required for the higher piston speeds.

In the Laidlaw-Dunn-Gordon compressors, a valve gear is used that mechanically controls the opening and closing of the suction and the closing of the discharge.

Effect of Altitude.—As the density of the atmosphere decreases with the altitude, a compressor located at a high altitude will take in a smaller weight of air at each stroke. The reduction of pressure at the inlet affects the power expended in compressing the air, but the decrease in power required does not vary in the same ratio

TABLE 10.—VOLUMETRIC AND HORSEPOWER COEFFICIENTS FOR TWO-STAGE AIR COMPRESSION

Altitude, feet	Barometer pressure, pounds per square inch	Terminal gage pressure, pounds per square inch													
		70		80		90		100		120		140		150	
		Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients	Horsepower coefficients	Volumetric coefficients
Sea level	14.72	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
1,000	14.17	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97	0.98	0.97
2,000	13.64	0.97	0.94	0.96	0.94	0.96	0.94	0.96	0.94	0.96	0.93	0.96	0.93	0.96	0.93
3,000	13.13	0.95	0.91	0.95	0.91	0.94	0.91	0.94	0.91	0.94	0.90	0.94	0.90	0.94	0.90
4,000	12.64	0.93	0.88	0.93	0.88	0.93	0.88	0.92	0.88	0.92	0.87	0.92	0.87	0.92	0.87
5,000	12.17	0.91	0.85	0.91	0.85	0.91	0.85	0.91	0.84	0.90	0.84	0.90	0.84	0.90	0.84
6,000	11.71	0.90	0.82	0.89	0.82	0.89	0.82	0.89	0.82	0.88	0.82	0.88	0.81	0.88	0.81
7,000	11.27	0.88	0.80	0.88	0.79	0.87	0.79	0.87	0.79	0.86	0.79	0.86	0.78	0.86	0.78
8,000	10.85	0.86	0.77	0.86	0.77	0.85	0.77	0.85	0.76	0.85	0.76	0.84	0.76	0.84	0.76
9,000	10.45	0.85	0.75	0.84	0.74	0.84	0.74	0.83	0.74	0.83	0.73	0.82	0.73	0.82	0.73
10,000	10.06	0.83	0.72	0.83	0.72	0.82	0.72	0.82	0.71	0.81	0.71	0.81	0.71	0.80	0.70
11,000	9.69	0.82	0.70	0.81	0.70	0.80	0.69	0.80	0.69	0.79	0.68	0.79	0.68	0.79	0.68
12,000	9.33	0.80	0.68	0.79	0.67	0.79	0.67	0.78	0.67	0.78	0.66	0.77	0.66	0.77	0.66
13,000	8.98	0.78	0.65	0.78	0.65	0.77	0.65	0.77	0.64	0.76	0.64	0.75	0.63	0.75	0.63
14,000	8.64	0.77	0.63	0.76	0.63	0.76	0.62	0.75	0.62	0.74	0.62	0.74	0.61	0.74	0.61
15,000	8.32	0.75	0.61	0.74	0.61	0.74	0.60	0.74	0.60	0.73	0.59	0.72	0.59	0.72	0.59

as the decrease in capacity. For this reason, compressors to be used at high altitudes should have the steam and air cylinders properly proportioned to meet the varying conditions at different levels. Table 10, published by the Sullivan Machinery Co., of Chicago, Ill., shows the variation in capacity and horsepower for various altitudes. The altitudes given are heights above mean sea level and are subject to correction for temperature and latitude. From the table it can be seen that for a two-stage compressor discharging at 100-lb. pressure when operating at an altitude of 8000 ft. the volumetric capacity will be only 76 per cent of that at mean sea level, while the horsepower required will be 85 per cent of that at mean sea level.

Efficiency.—In practice it is found that compressors with mechanically operated rotary inlet valves show volumetric efficiency varying from 91 per cent at 100 r.p.m. to 88 per cent at 188 r.p.m. Piston inlet machines at 100 r.p.m. give 88 per cent efficiency, and at 188 r.p.m., 79 per cent.

Centrifugal Compressors.—Centrifugal compressors differ from centrifugal pumps (see page 432) only in handling gases instead of liquids, and are similarly classified as regards the number of inlets per impeller, the direction of the impeller tips at impeller exit, and the number of stages employed. They are further classified into low-pressure (1 to 5 lb. per sq. in.) and high-pressure (above 5 lb.) compressors, and also into the radial-inlet type and the axial-inlet type, according as the gas enters at right angles to the shaft or in a direction parallel to the shaft.

Centrifugal compressors for pressures below 1 lb. per sq. in. are generally known as blowers or centrifugal fans (see page 441); in these the kinetic energy of the gas at the impeller exit is usually allowed to dissipate itself in eddies. For air pressures of 5 lb. per sq. in. and under, a single impeller is generally sufficient. For comparatively light gases, however, a pressure of 5 lb. may require two or more impellers in series, or a multistage compressor. Such a compressor is also frequently spoken of as a high-pressure gas compressor. For quantities of gas of 10,000 cu. ft. per min. and over, compressors of the radial-inlet type require shrouds or reinforcing rings at the inner ends of the impeller blades to prevent the wide blades from crumpling at the inlet under the action of centrifugal stresses. Impellers of the axial-inlet type are not subject to such crumpling and are therefore generally used when large volumes are handled. The radial-discharge impeller is the one best adapted for high peripheral speeds, and is the type most commonly used. Both the backward-discharge and the forward-discharge impellers require shrouds at their outer peripheries. The former are frequently resorted to when large gas quantities (requiring a large impeller inlet) are to be raised to a comparatively low pressure with a direct-connected high-revolution-per-minute driver; the latter are but rarely used. Multistage compressors are usually provided with special means for cooling the gas during its passage through each impeller and from stage to stage, and also for preventing leakage from stage to stage and to the atmosphere.

The centrifugal compressor occupies comparatively little room for its output; direct connected to an electric motor or to a steam turbine, it forms a very compact unit. Besides its bearings, it has no rubbing or wearing parts; it contains no moving valves or springs; and it requires a minimum of attendance and oiling. It is also fairly free from vibrations and requires comparatively light foundations. At constant revolutions per minute it will maintain approximately constant pressure for widely varying quantities of gas, which makes it very desirable for general power transmission. If no gas is required temporarily, the discharge pipe may be shut off without stopping the compressor or wasting the gas into the atmosphere. Its steadiness of blast makes it also very valuable for oil burning and for general forge work.

For blast furnaces, where the resistance to the flow of the air is likely to vary from time to time, while a uniform supply of oxygen or air regardless of such variation in resistance is important, the centrifugal compressor may be supplied with a constant-volume governor. This governor, actuated by the variation of air velocity in the compressor inlet, causes the speed of the driver to vary in accordance with the needs of the furnace, and thus maintains a constant volume (referred to atmospheric conditions) of air against widely varying pressures with fairly constant efficiency.

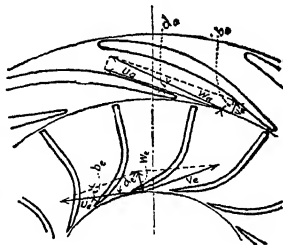


FIG. 15.

The centrifugal compressor also finds wide application as an exhauster in ash conveying, sawdust conveying, and in the general pneumatic conveying of coal, cement, rice, starch, etc. For intermittent work, such as pneumatic cash and mail conveying, its prompt response to overloads allows the use of a comparatively small driver. In coke-oven-gas manufacture the centrifugal compressor maintains constant suction on the gas main, and then compresses the gas so that it will flow through the condensers, purifiers, and into the gas holders. It is frequently used as a booster to a high-pressure reciprocating compressor, the two compressors forming together a very compact and efficient set. By compressing the air, say, to 30 lb. per sq. in. gauge in the centrifugal compressor, the volume of the air to be handled by the reciprocating compressor is only about one-third of what it would otherwise be, making it possible to employ a much smaller unit.

While a centrifugal compressor will maintain a fairly constant pressure over a wide range of quantities, there is for every speed a certain range of quantity at which the discharge vanes cease to cooperate, causing a sudden drop in pressure. This "breakdown" region can be pushed back toward lighter loads, and the drop in pressure made less abrupt by making the discharge vanes very few and their inlet angle small. Also, when working on that part of the pressure curve where the pressure increases with the quantity or remains constant, there are usually pressure surges or pulsations which, while slight in themselves, may be greatly intensified by a sort of resonance effect if the volume of the inlet and of the discharge piping happens to have a certain critical value. A slight throttling of the inlet will always stop these pulsations by making the pressure curve slightly drooping.

CHARACTERISTIC CURVES

Uses of Characteristics.—Besides affording smooth curves of hydraulic and shaft efficiencies, a set of characteristic curves as shown in Fig. 16 enables one readily to

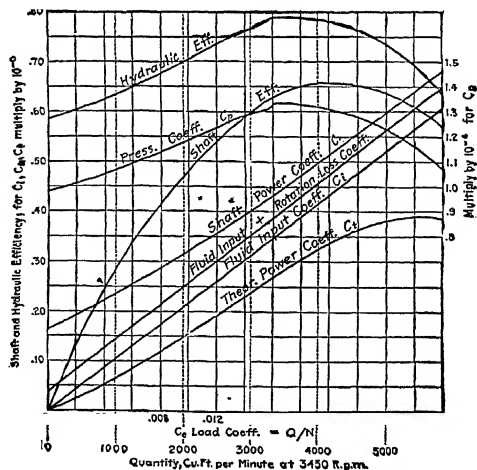


FIG. 16.—Characteristic curves of centrifugal compressors.

draw reliable pressure and power curves against quantity for any given wheel speed or revolutions per minute. In that case, the load-coefficient scale may be replaced

by a quantity scale, while the readings of the C_p and the C_s curves will give the data for the corresponding pressures and powers. (In Fig. 16 the wheel speed has, in all the coefficients, been replaced by the revolutions per minute.) The system of characteristic curves is found to hold true for various sizes of centrifugal compressors, centrifugal blowers, and centrifugal pumps, giving in each case consistent curves regardless of the actual speeds, pressures, and powers. Only in ventilating-fan blowers do there seem to be occasionally some serious discrepancies near the point of maximum efficiency.

Multistage Centrifugal Compressors.—A multistage compressor consists of a number of single-stage compressors connected in series. When the number of single-stage compressors is small, they are usually enclosed in the same casing; but when the number exceeds eight or ten, they are generally subdivided between two or more casings, separated by intercoolers. Multistage compressors are frequently built with single-inlet impellers, depending for the overcoming of the axial thrust on balancing pistons, on special grouping of the impellers, and on similar devices, also with backward-discharge impellers, when they are supplied with substantial shrouds to prevent the blades from bending or breaking under the action of centrifugal force. They are also built with all the impellers of the double-inlet type, thus obviating the need of all balancing means and requiring a much smaller number of impellers; the interstage passages, however, become rather complicated.

The successive impellers in a multistage compressor, handling smaller and smaller volumes of air, should be designed on the principle of similar compressors if all stages are to be equally efficient. Since the revolutions per minute is the same for all impellers, the diameters and all the other dimensions should vary *inversely as the cube root of the density*. In practice, however, the impellers are divided into two or more groups, and each group designed for its own average conditions.

Cooling.—The cooling of the gas during its passage through a multistage compressor is of paramount importance for high efficiency and low power consumption, and ample passages for cooling water must be provided in the diaphragms between the stages. For pressures below 50 lb. per sq. in., it is generally aimed to keep the temperature down to that corresponding to adiabatic compression; while for higher pressures, isothermal compression is more usually aimed at by the introduction of intercoolers between groups of stages. In the latter case, the hydraulic efficiency is given on the basis of isothermal compression.

Theoretical Power.—The theoretical horsepower required to compress adiabatically and deliver 100 cu. ft. of gas per minute (initial pressure, p_1 lb. per sq. in.) is $\text{hp.}_a = 1.501 p_1 [(p_2/p_1)^{0.29} - 1]$; and for isothermal compression it is $\text{hp.}_i = 1.004 p_1 \log_{10} (p_2/p_1)$. Table 11 gives the ratio of the theoretical work for isothermal compression to that for adiabatic compression for the largest range of pressures likely to be met with in practice.

TABLE 11

p_2/p_1	1.5	2.0	2.5	3	4	5	6	7	8	9	10
(Isothermal ÷ Adiabatic).....	0.940	0.904	0.875	0.85	0.812	0.784	0.763	0.744	0.728	0.715	0.703

Leakage.—To reduce the leakage between stages and from the inlet of the last stage to the atmosphere, labyrinth packings are usually provided. In general, the leakage may be assumed as about 3 per cent of the rated quantity, about two-thirds of this taking place between the inlet of the last stage and the atmosphere. This loss is fairly independent of the number of stages.

The efficiency of a centrifugal compressor is the ratio of the theoretical power, adiabatic or isothermal, corresponding to the rated quantity and pressure, divided by the power delivered to the compressor shaft. There is, however, considerable room for controversy and misunderstanding in this connection, owing to the difficulty of measuring accurately the output of the driver. The input of the driver, however, is generally easily determinable. It is, therefore, customary to specify that the input of the driver per 100 cu. ft. of air delivered by the compressor shall not exceed so many kilowatt-hours, or so many pounds of steam, the steam conditions being those of the customer's plant.

Commercial sizes of the single-stage compressors have pressure ratings from 0.75 to 4 lb. per sq. in. and capacities from a lower limit of 500 cu. ft. per min. to a higher limit which ranges from 12,000 cu. ft. at 0.75-lb. pressure down to 3000 cu. ft. at 4-lb. pressure. The multistage compressors are built in the following sizes:

Cubic feet per minute.....	4,500	9,000	16,000	25,000	40,000	50,000
Pressure, pounds per square inch.	6-35	6-25	6-25	12-30	12-30	12-30

Multistage compressors have also been built for pressures as high as 150 lb. per sq. in., the volumes being comparatively small.

With a driver allowing speed variation, like a steam turbine or direct-current motor, a centrifugal compressor of a given rating will operate with a number of different combinations of pressure and volume without excessive loss in efficiency. The different rated pressures of multistage compressors are usually obtained by the use of a different number of stages, all stages in a given size of machine being similar. A given compressor may, however, operate successfully between, say, 15 and 30 lb. per sq. in., if the driver will allow of the necessary speed variation.

Without any special governing device a centrifugal compressor driven at constant speed will maintain a substantially constant pressure, regardless of the volume of air delivered. Should there be temporarily no demand for air while the compressor is being driven at full speed, the casing will become slightly hotter, but otherwise the machine will not be subjected to any particular strains. The power required by a centrifugal compressor between half load and, say, 25 per cent overload varies approximately as the load. The following table shows the usual variation of pressure and power with the volume handled by a single-stage compressor at constant speed.

For blast-furnace purposes, where a constant quantity of air is to be delivered against varying pressures, a constant-volume governor has been developed.

CHARACTERISTICS OF CENTRIFUGAL COMPRESSORS AT CONSTANT SPEED

Volume, per cent of full load.	0	20	40	50	60	70	80	100	120
Pressure, per cent of full load.	92	94	97.5	100	103	105	105	100	91
Power, per cent of full load. . .	50	53	58	62	66	73	81	100	120

BLOWERS AND COMPRESSORS

Rotary blowers are built for air pressures varying from 6 oz. to 10 or even 12 lb. per sq. in. The best efficiencies of this type of blower, however, are usually secured below 5-lb. pressure, but the simplicity of the machine gives it an advantage over compressors of the piston type and frequently warrants its installation for the higher

pressures indicated when designed for this purpose. As the machine operates by displacement, it is usually preferred for cupola practice, and for copper and lead blast furnaces, because its positive action will not permit a reduction in air supply if the cupola tends to clog. For other uses of air at pressures below 8 oz., the fan is ordinarily more economical.

Blowers of this type may be designed to give either constant volume or constant pressure, and to handle either liquids or gases. They consist of a casing containing one or more revolving impellers of various forms of design. Figure 17 represents a cross section of the Sturtevant high-pressure blower, which is built in capacities ranging from 5 to 15,000 cu. ft. per min. at 8-oz. pressure (speeds, from 375 to 800 r.p.m. for the smallest size down to 160 to 220 r.p.m. for the largest; weights with subbase range from 200 to 39,000 lb.). The smaller machines have a vertical arrangement of shafts, while the larger types usually have their shafts in the same horizontal plane. Two impeller blades are always in action, and leakage by one is caught by the other. The proper size of blower for a cupola may be calculated on the basis of 30,000 cu. ft. of air per ton of iron melted. Sturtevant machines are also used for handling gases. The capacities for the various sizes of gas exhausters range from 7500 to 900,000 cu. ft. per hr. at 8-oz. pressure, making no allowance for shrinkage, which will vary from 10 to 20 per cent, depending on the gas and its pressure. The inlet and outlet diam-

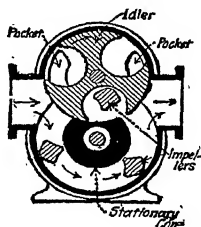


FIG. 17.—Sturtevant rotary blower.

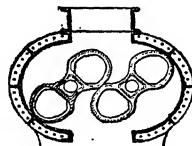


FIG. 18.—Roots blower.

eters run from 3 in. in the smallest size to 30 in. in the largest; weights, from 400 to 33,000 lb.

Figure 18 illustrates a cross section of a Roots blower. The two impellers are symmetrical and are driven in opposite directions by gears outside the casing. The impellers do not touch each other nor the casing, but the clearance is reduced to a minimum in order to reduce slip or leakage. The amount of this slip or leakage may be determined by operating the machine with a closed discharge, at a speed sufficient to maintain the required discharge pressure. The amount is usually largest in machines of smallest capacity, *i.e.*, a machine displacing 0.75 cu. ft. per revolution at a pressure of 1 lb. will have a slip of 60 to 70 revolutions, while a machine having a capacity of 300 cu. ft. per revolution will have a slip of 3 to 5 revolutions. For intermediate capacities, the slip will vary proportionally and increase with higher pressure as the square root of the discharge pressure, *i.e.*, at 4-lb. pressure the slip will be approximately twice that at 1 lb.

In most blower work the so-called hydraulic formula for horsepower will be found satisfactory; horsepower = $Q(p_2 - p_1)/33,000$, where Q is the cubic feet of air compressed per minute, p_1 the initial pressure, and p_2 the final pressure in pounds per square foot. To get the actual horsepower at the shaft, the horsepower should be divided by the efficiency, which will vary from 0.60 to 0.80. If the speed remains unchanged, owing to the small clearances, the horsepower tends to increase tremendously with decrease in orifice.

The so-called Hytor compressor (or exhaust) works on the principle of a number of columns of liquid acting as pistons between the rotor teeth. The casing is ovoid, and as the moving liquid swings away from the rotor shaft the gas is drawn into the space vacated. As the liquid swings back further in the revolution, the gas is compressed and tends to escape through the outlets.

The efficiency is high, but the effluent gas will always be saturated with the vapor of the liquid used as the circulating medium.

Air Drying.—This can be done by contact with dehydrating (usually deliquescent) salts or acids, by compression and cooling, by refrigeration, by adsorption.

The best salts and acids are KOH, NaOH, H_2SO_4 , CaCl_2 , CaBr_2 , MgCl_2 , ZnCl_2 , ZnBr_2 , and P_2O_5 . If CaCl_2 , CaBr_2 , or the like are used, the reagent should usually not be allowed to deliquesce, but only to hydrate partially, as it requires less fuel per pound of water to dry the lower hydrates than the higher. MgCl_2 cannot be completely dried.

Sulphuric acid drying requires the use of 60 per cent acid or stronger. As it is impossible to get good contact without the use of sprays, there is danger of introducing an acid mist which may seriously affect the process.

Drying by compression is prohibitive in cost unless compression is incidental to some other process.

Drying by refrigeration is probably the best method for metallurgical work. Lowering the temperature of any gas causes condensation of the excess of the vapor in the air at its original temperature over its dew-point content at the lower temperature.

Chemical reagents are a costly method of drying, but a convenient one for small installations, while calcium chloride was for some time used as a dehydrating agent on a large scale for the blast to an iron furnace.

TABLE 12.—WATER VAPOR IN AIR¹

Temperature, degrees Fahrenheit	Vapor pressure, inches Hg	Grains per cubic foot	Volume displaced, per cent
-40	0.0039	0.072	0.024
-20	0.0126	0.166	0.042
-10	0.0222	0.285	0.074
0	0.0383	0.481	0.128
+10	0.0631	0.776	0.211
20	0.1026	1.235	0.343
30	0.164	1.935	0.548
40	0.247	2.849	0.825
50	0.360	4.076	1.203
60	0.517	5.745	1.728
70	0.732	7.980	2.446
80	1.022	10.934	3.416
90	1.408	14.790	4.706
100	1.916	19.766	6.404
110	2.576	26.112	8.610

¹ Table by HOLDEN, EDWIN C., from *Methods of Air Drying*, *Chem. & Met. Eng.*, Vol. 28, p. 801, May 7, 1923.

From Table 12 it is evident that, to refrigerate to 2 grains of water per cubic foot, the temperature must be reduced to 30°F. J. E. Johnson, Jr., in "Blast Furnace

Construction," states that to refrigerate 1000 cu. ft. of air per minute from 85°F. and 70 per cent saturation to 25°F., with condenser temperature at 85°F., will require

With single-stage refrigeration	11.56 hp.
With two-stage refrigeration	7.95 hp.
With two-stage refrigeration and regeneration	4.97 hp.

Edwin C. Holden comments on the above that each refinement adds to the cost of the plant, and the regenerating towers required in the last scheme add seriously to the power required to blow the air blast through the system, so that the ultimate relative economy is not what it appears from the figures given.¹

If air is to be refrigerated below 32°F., as a matter of practical working it should be done in two stages, one to about 33 or 34°F., removing the bulk of the water as a liquid, then another stage to the final temperature. Otherwise, there is a large deposit of ice on the pipes that cuts down efficiency and is costly to remove. It is easier to remove rain than snow from a sidewalk.

Removal of moisture from air by adsorption may be done by passing the air over adsorbents like fuller's earth, bauxite, alumina, charcoal, or silica gel. It is estimated that with a three-stage silica-gel adsorption apparatus the power per 1000 cu. ft. per min. flow will be 70 to 90 kw.-hr., considerably more than refrigeration.

The preheating of gases is seldom practiced in nonferrous metallurgy. In certain difficult roasting operations the air is preheated by moving it through pipes running through exhaust-gas flues, while the old Pearce furnaces at Argo, Colo., had a tortuous system of passages running under the reverberatory bottom which cooled the bottom and preheated the air. Some of the earlier pyritic furnaces had tops in which the exhaust gases passed around the blast mains. In general, however, the question of recuperators, blast stoves, and regenerative reverberatories is left to the ferrous metallurgist, and information should be sought in the ferrous metallurgies.

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¹ *Loc. cit.*

CHAPTER XV

MATERIALS OF METALLURGICAL CONSTRUCTION

BY JOHN A. CHURCH¹

Scope of Chapter.—This section deals with those properties of materials which govern selection and use of those materials (for plant layout, see Chap. XVI). The subdivisions are Review of Structural Theory; Engineering Materials (for general structural purposes); Materials of Metallurgical Equipment (for withstanding high temperatures, or chemical corrosion, or both; for refractories see Chap. XI).

REVIEW OF STRUCTURAL THEORY

This review is intended to cover only simple problems of design. For complete theory and practice see the bibliography; for complex problems refer to a competent structural engineer.

STRUCTURAL MEMBERS

Strength Properties.—Load applied to a structural member causes change of form (*strain*), which is resisted within the member by forces of cohesion (*stresses*). Sustainment of the load implies equilibrium of external and internal forces, whence load equals and measures stress. Deformation may occur in three ways: by stretching, compression, and slip (of one particle past the next, alongside), resisted, respectively, by *tensile*, *compressive*, and *shearing* stresses; total resistance of the member in tension and compression (*direct stress*) depends on its sectional area normal to—in shear parallel to—the direction of stress. *Unit stresses* are averages per unit of such sectional area (before deformation); they do not imply uniform distribution of stress, though used to measure its intensity over limited areas. For a limited range of stress the ratio of stress to strain is a constant (*modulus* or *coefficient of elasticity*) characteristic of the material; within this range, deformation is not permanent, for zero stress implies zero strain—*i.e.*, the material is elastic. The *elastic (proportional) limit* marks the upper end of the range; at higher stresses strain is accelerated and persists after removal of load; *i.e.*, the material tends to become *plastic*.

Tension.—If plasticity is well defined, there is a *yield point* slightly above the elastic limit at which, for small stress increments, strain becomes excessive. Further loading develops maximum stress (*ultimate strength*), measured by maximum sustained load, above which failure begins by local constriction (*necking*) and strain increases so fast that the load may even be partially relieved before the neck finally parts. A brittle material shows an ill-defined yield point and breaks sharply with little necking.

Compression exhibits a like elastic range and limit, and for many materials the modulus of elasticity is the same as in tension. Beyond the elastic limit, brittle materials develop their ultimate compressive strengths and then fail by crumbling or breaking; plastic materials continue to deform, therefore show no true ultimate strength and no definite failure.

Shear.—Direct tension causes not only stretch (tensile strain) but also lateral contraction, due to slip along planes oblique to the direction of load—*i.e.*, shearing

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strains, resisted by shearing stresses acting *along* those planes. *Poisson's ratio* g of shearing to tensile strain varies with the material, theoretically within the limits of 0 and 0.5. Ratio of shearing to tensile modulus of elasticity is approximately $1:2(1 + g)$, or from 0.33 to 0.5, roughly confirmed by test. Unit shearing stress varies from 0 on planes parallel or normal up to 0.5-unit tensile stress on planes at 45 deg. to direction of load. Direct compression also involves shear, with lateral expansion replacing contraction, but similar relationships. Shear has its own elastic limit and ultimate strength, which under certain conditions may govern design.

Systems of Loading.—The external forces acting on a structural member comprise active loads and reactive forces of support. If collinear, they produce direct tension or compression; if parallel and coplanar, but not collinear, *bending*; if parallel but not coplanar, *torsion*, with or without bending.

In Fig. 1, AB is the axis of a member subject to vertical forces P_1, P_2 coplanar with AB , and to P_3 vertical but not coplanar. On normal d from P_3 at the axis, apply the balanced forces P_4, P_6 , each equal and parallel to P_3 ; they do not affect equilibrium.

P_5 are coplanar and exert (1) a pure bending effort, measured at any point

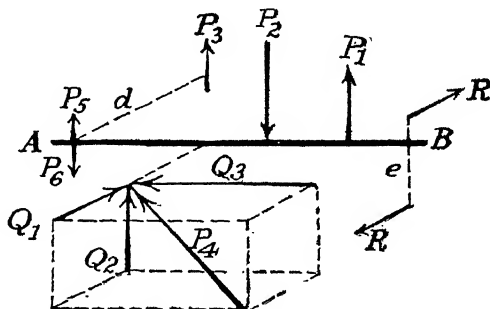


FIG. 1.—Forms of loading.

on AB by the algebraic sum of their moments on either side about that point; (2) an effort of transverse shear on planes normal to AB . P_3, P_6 form a couple exerting an effort of torsion, measured by the torsive moment P_3d . If the system is static, there is an equal and opposite reactive couple at the support, whose moment $Re = P_3d$. Any oblique force P_4 is resolved at point of application (shown here in the horizontal plane through AB) into components Q_1 horizontal, Q_2 vertical, and Q_3 parallel to AB . Q_1 with its reactions at the supports (not shown) exerts horizontal bending and shearing efforts. The total bending moment and transverse shear at any point on AB is, then, the square root of the summed squares of the separate moments and shears due to P_1, P_2, P_5 and Q_1 . Q_2 , not coplanar with AB , is analyzed like P_3 . Q_3 exerts a horizontal bending effort and presents the common case of an eccentric longitudinal load. These principles serve to analyze any system of external forces.

Bending—Beams.—The simplest case is a slender horizontal member (*beam*) under gravitational load; *i.e.*, all external forces are vertical. Premises: (1) a straight beam of uniform section symmetrical to the plane of external forces; (2) all stresses within the elastic limit; (3) the bending strain (*deflection*) small; (4) elastic moduli alike in tension and compression.

Stress Moment.—Under load the beam is slightly curved; the external bending moment is resisted by an equal moment of horizontal stresses, compressive on the concave, tensile on the convex side—hence zero at some intermediate section (the

neutral surface) whose intersection with the plane of symmetry is the *neutral axis*. For equilibrium the sum of tensile and compressive stresses must be zero. Tests show that unit strain—hence stress intensity—varies directly with distance z from the neutral surface (Fig. 2). If dA is an element of sectional area and S the stress intensity at distance e (*extreme fiber stress*), then stress on $dA = \frac{z}{e} S dA$, and total stress on the section $= \frac{S}{e} \int_{e_1}^e z dA = 0$ (as above), whence $\int_{e_1}^e z dA = 0$ and the neutral axis cuts the centroid of the section. The stress moment about AB equals

$$\frac{S}{e} \int_{e_1}^e z^2 dA = \frac{SI}{e} = M \quad (1)$$

wherein I is the moment of inertia of the section about AB and M is the external bending moment. Bending increases the differential fiber length $mn (= ab)$ by np , whence unit tensile strain $= \frac{np}{mn}$ " " $\frac{S}{E}$ the elastic modulus), and

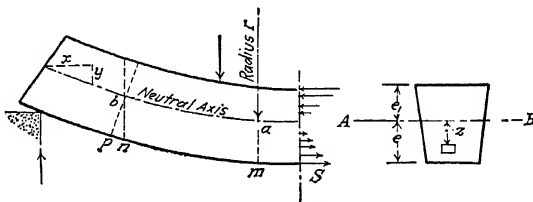


Fig. 2.—Bending under stress.

$M = \frac{SI}{e} = EI \frac{d^2y}{dx^2}$; the deflection y due to bending is computed by expressing M in terms of x (see below). For I and e see Table 1 (general). I/e is the *section modulus*.

External Moments.—A simple beam is one that rests freely on end supports (Fig. 3). If $Q_1 Q_2$ are the reactions due to P_2 alone, $Q_1 + Q_2 = P_2$ and moments $Q_1 l - P_2 a = Q_2 l - P_2(l - a) = 0$, whence Q_1 and Q_2 are computed. Moment at X due to $P_2 = Q_1(l - x) = Q_2 x - P_2(x - a)$, maximum for $x = a$; if fe scales this maximum, ordinates to ced scale moments due to P_2 throughout. If ckd is the corresponding diagram for P_1 alone, draw $em =$ intercept ji , and kn in like manner; then ordinates to $cnmd$ scale moments due to P_1 and P_2 combined. Maximum deflection due to any single load P_2 is $\frac{3EI}{384} \left[\frac{a^2}{3} \right]^3$. A given load produces maximum moment

and deflection when concentrated at span center (see Table 2). Total moments due to any combination of loads, uniform and concentrated, are found by simple addition of ordinates in the moment diagrams shown in Fig. 3 and Table 2.

Shear.—In Fig. 3, ordinates in diagrams $hgvw$ represent vertical shear V ; $hg = R_1$, $vw = R_2$, $os = P_1$, $rz = P_2$; shear passes through zero where the moment is maximum; its intensity is V/BH (B and H are width and depth of beam). There is an equal horizontal shear in the beam, zero at the extreme fibers, of maximum intensity v at the neutral surface; $v = NV/BH$ ($N = \frac{3}{2}$ for a rectangular or $\frac{4}{3}$ for a circular section). This longitudinal shear may control in designing short, deep wooden beams, whose strength is slight along the grain: a simple rectangular beam of this sort, with uniform

load, is proportioned to allowable limits of stress by $v/S = H/l$; with single center load, by $v/S = H/2l$. Deflection due to vertical shear is important only in beams wherein $l = 10H$ or less. For a rectangular section, it approximates $C(Pl/GBH)$ (G = the elastic modulus in shear); C is 0.3 for a simple beam with single center load, 1.2 for a cantilever with single end load, and one-third of these values for uniform loading ($P = pl$).

Constrained Beams.—A constrained beam is so held that the support applies a reverse moment and at some point of *contraflexure* reverses the curvature due to bending; at this point the moment is zero. A *cantilever* is supported at one end only (see Table 2). A *continuous* beam extends without break over several supports. It is stiffer than a series of simple beams of like spans and section, but subject to serious changes in computed stresses due to minor readjustments of support.

Overstressed Beams.—Beyond the elastic limit, fiber stress and strain are no longer proportional and Eq. (1) does not apply. Extreme fiber stress at failure (computed from the breaking moment) is termed the *modulus of rupture*, an experimental index of strength.

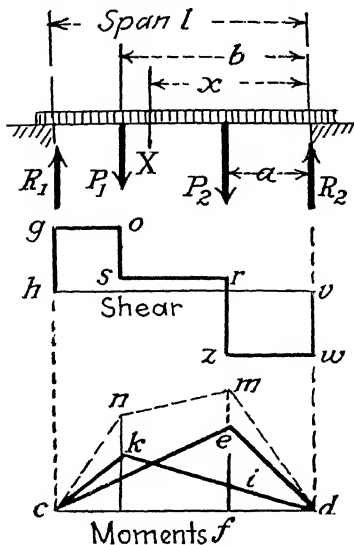


FIG. 3.—Stress in beams.

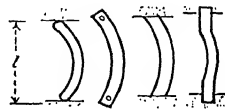


FIG. 4.—Loaded columns.

Bending—Columns.—A column under axial load fails by lateral bending before its full compressive strength is developed. If r_0 is the least radius of gyration ($= \sqrt{I/\text{area}}$) of the column section, a slenderness ratio l/r_0 above 120 defines a *long column*, less than 120 a *short column*. The conditions of the ends (Fig. 4) influence column strength, but can rarely be exactly prescribed by the designer; i.e., pin ends are modified by frictional restraint, fixed ends by the practical difficulty of rigid support—hence the empirical nature of many formulas.

Euler's formula (for long columns), viz., total axial load causing failure by pure bending $= n\pi^2 EI/l^2$, wherein $n = 2.5$ for square ends and 1.67 for pin ends in practical use, is applicable only when the result is not over one-third ultimate compressive strength.

Rankine's formula (for short columns) is safe load in pounds per square inch for combined compression and bending $= \frac{k \times \text{area}}{1 + \frac{C \times \text{area}}{N}}$, wherein k is allowable unit com-

pressive stress in pounds per square inch and C and N are constants (Table 3).

Straight-line Formulas (Empirical).—General form: Safe load in pounds per square inch $= k - C \frac{l}{r_0^2}$, wherein k is allowable unit compressive stress and C is a

TABLES 1 AND 2

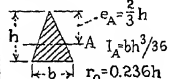
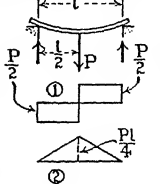
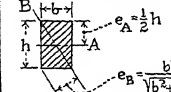
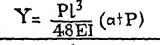
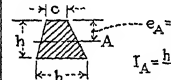
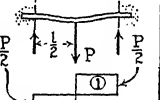
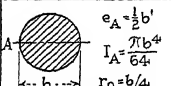
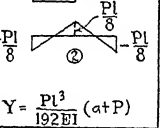
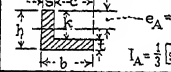
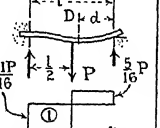
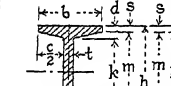
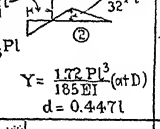
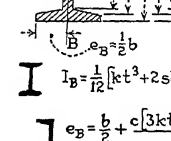
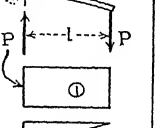
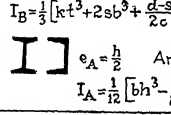
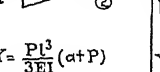
<p>Table 1. Properties of Sections Axes through centroids of sections. I = Moment of inertia r_0 = Radius of gyration e = Distance from axis to extreme fiber</p>	<p>Table 2. — Formulas for Beams Load (P) in lbs., (p) in lbs. per inch. (l) in inches, (I) in inches⁴, (E) in lbs. per inch². Max. deflection due to bending (Y) in inches. ① Vertical shears, in lbs. ② Moments, in inch-lbs.</p>
 <p>$e_A = \frac{2}{3}h$ $I_A = \frac{bh^3}{36}$ $r_0 = 0.236h$</p>	 <p>① $\frac{P}{2}$ ② $\frac{Pl}{4}$</p>
 <p>$e_A = \frac{h}{2}$ $I_A = \frac{bh^3}{12}$ $r_0 = 0.289h$</p>	 <p>① $\frac{pl}{2}$ ② Parabola $\frac{pl^2}{8}$</p> <p>$M_x = \frac{Px^2}{2}(l-x)$ $Y = \frac{5pl^4}{384EI} \left(\text{at } \frac{l}{2} \right)$</p>
 <p>$e_A = \frac{h}{2}$ $I_A = \frac{bh^3}{12}$ $r_0 = 0.289h$</p>	 <p>① $\frac{P}{2}$ ② $\frac{Pl}{4}$</p>
 <p>$e_A = \frac{1}{2}b'$ $I_A = \frac{\pi b^4}{64}$ $r_0 = b/4$</p>	 <p>① $\frac{P}{2}$ ② $\frac{Pl}{4}$</p>
 <p>$e_A = \frac{h}{2}$ $I_A = \frac{bh^3}{12}$ $r_0 = 0.289h$</p>	 <p>① $\frac{P}{2}$ ② $\frac{Pl}{4}$</p>
 <p>$e_A = \frac{h}{2}$ $I_A = \frac{bh^3}{12}$ $r_0 = 0.289h$</p>	 <p>① $\frac{P}{2}$ ② $\frac{Pl}{4}$</p>
 <p>$e_A = \frac{h}{2}$ $I_A = \frac{bh^3}{12}$ $r_0 = 0.289h$</p>	 <p>① $\frac{P}{2}$ ② $\frac{Pl}{4}$</p>
 <p>$e_A = \frac{h}{2}$ $I_A = \frac{bh^3}{12}$ $r_0 = 0.289h$</p>	 <p>① $\frac{P}{2}$ ② $\frac{Pl}{4}$</p>

TABLE 3
(After Merriman)

End condition	C	Material	N
Both fixed.....	1	Timber.....	3,000
Both round.....	4	Cast iron.....	5,000
One fixed, one round.....	1.95	Wrought iron.....	36,000
		Steel.....	25,000

constant, both in pounds per square inch. For specific applications see text on cast iron, steel, and timber.

Moving Loads.—The weight of a locomotive is treated as a series of *wheel concentrations*, the weight of a train as a uniform load or a series of equal concentrations uniformly spaced. Required—values of, and positions of loading for (1) maximum moments and shears at some specific section of a beam so loaded; (2) absolute maxima for the beam.

Two loads (i.e., traveling crane truck), maxima for the beam (Fig. 5a): If C is the resultant of the two loads, maximum moment occurs under the heavier when $r = s$; for two *equal* loads, under that nearer span center when $s = \frac{1}{4}a$ (unless a exceeds $0.586l$, in which case the maximum is $Pl/4$, wherein $P =$ either load).

Any number of loads, criteria for section A (Fig. 5b): Moment at A is maximum when span center bisects the distance from A to the resultant of the loads, or when $\Sigma Q/\Sigma P = m/l$ (wherein ΣP is the

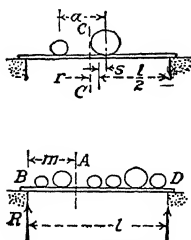


FIG. 5.—Moving loads.

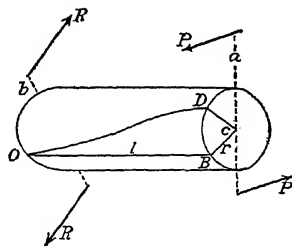


Table 4 — Torsion

Section	Torsive Moment (TJ)÷r
	$\frac{\pi r^3}{2} \cdot T$
	$\frac{\pi a b^2}{16} \cdot T$
	$\frac{\pi (a^4 - s^4)}{2r} \cdot T$
	$\frac{a^2 b^2}{3a + 1.8b} \cdot T$

FIG. 6.—Torsion.

sum of loads on the span and ΣQ of those on BA); usually true only when ΣQ includes a part load (i.e., a wheel is over A). A long series of loads may have several maxima, one of which is the absolute (found by trial). Maximum shear at $A = R - \Sigma Q$ and occurs when some load (found by trial) is over A but excluded from ΣQ in computation. Maximum shear in the beam is R , itself maximum when some load (found by trial) is over B but included in ΣP .

Uniform Load.—Maximum moment at any section requires a fully loaded span; maximum shear occurs when either BA or DA (whichever is greater) is fully loaded and the rest of the span without load.

Torsion.—In Fig. 6, the couple Pa acts on a cylindrical member of radius r and length $l (= OB)$ and induces an equal reactive couple Rb at the support. These couples tend to rotate each normal circular section about the axis of the member so that surface points originally on OB take positions OD ; this torsive deformation is resisted by shearing stresses acting *along* these normal sections, and the external moment Pa is opposed by an equal moment of shearing stress. Assumptions: normal sections originally plane remain so in torsion; stresses are within the elastic limit; stress intensity varies directly with radial distance from the centroid of section C , where it is zero (true for circular, approximate for rectangular sections). Then if the stress intensity at periphery is T , that on any element dA of the section at distance z is $\frac{z}{r} T$, the total stress on this element is $\frac{z}{r} T dA$ and the stress moment $\frac{z^2}{r} T dA$; the total stress moment of the section is

wherein J is the polar moment of inertia of the section about the cylindrical axis.

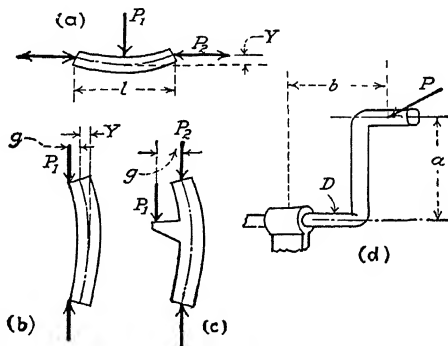


FIG. 7.—Eccentric column loads.

Total torsive strain is measured by the arc BD ; unit strain by $BD/l = T/G$ (wherein G is the elastic modulus in shear); angle of torsion BCD in radians $= BD/r = lT/Gr = lPa/GJ = 57.3 (lPa/GJ)$ in degrees (see Table 4).

Combined stress is the stress due to more than one type of loading—direct tension or compression, cross-bending, column loading (axial or eccentric), and torsion. If a superimposed load produces stress unlike the stress due to the member's own weight, combined treatment is always necessary.

Notation.— S = maximum unit extreme fiber stress due to bending, in pounds per square inch; e , I , E , and Y as in Eq. (1) and Table 2; Z = unit tensile or compressive stress due to axial load, in pounds per square inch; M_1 = bending moment due to bending load P_1 , in inch-pounds; A = sectional area of member in square inches; and l = its length in inches. The treatment of combined direct and bending loads follows Ketchum.

Cross-bending and Axial Load.—Table 2 shows that for either uniform or center load $Y = Pl^2/cEI$ and $M = Pl/k$, wherein c and k are numerical coefficients (for

uniform load $P = pl$; then $Y = Ml^2/nEI$, wherein $n = c/k$. In Fig. 7a the maximum bending moment M (causing deflection Y) = $M_1 \pm P_2Y = M_1 \pm \frac{P_2 M l^2}{nEI} = \frac{SI}{e}$; then

$$S = \frac{M_1 e}{I \pm \frac{P_2 l^2}{nE}} \quad (3)$$

Use the plus sign when P_2 is tensile, the minus when compressive; n is 10 for a simple beam, 24 for a beam fixed at one end and hinged (simply supported) at the other, 32 for a beam fixed at both ends. The maximum combined stress is

$$Z + S = \frac{P_2}{A} + \frac{M_1 e}{I \pm \frac{P_2 l^2}{nE}} \quad (4)$$

tensile in the lowermost fibers (with plus sign) when P_2 is tensile, compressive in the uppermost fibers (with minus sign) when P_2 is compressive.

Eccentric Column Loads.—For a single eccentric column, load P_1 (Fig. 7b) Eq. (4) becomes

$$Z + S = \frac{P_1}{A} + \frac{P_1 g e}{I - \frac{P_1 l^2}{nE}} \quad (5)$$

For two loads (Fig. 7c), P_1 eccentric and P_2 axial (*i.e.*, a post with side bracket supporting a crane girder), $Z + S = \frac{P_1 + P_2}{A} + \frac{P_1 g e}{I - \frac{(P_1 + P_2) l^2}{nE}}$

Rectangular I bar in tension of width B , depth H , length l , weight p per inch of length and w per cubic inch, under tensile axial load P (linear dimensions in inches, loads and weights in pounds). Here M_1 (due to own weight) = $pl^2/8$, $p = BHw$, $e = H/2$, $I = BHl^3/12$, $Z = P/A = P/BH$, and $P = ZBH$; then Eq. (3) takes the convenient form

$$S = \frac{wH}{\frac{4H^3}{3l^2} + \frac{8Z}{5E}} \quad (6)$$

If the bar is an inclined member, multiply this value of S by the sine of its angle to the vertical. Maximum combined stress is $Z + S$ as before.

Bending and Torsion (as in a Crankshaft).—Treat the horizontal thrust P in Fig. 7d as in the case of P_2 , Fig. 1 (*q.v.*); design the shaft D to resist torsive moment Pa [Eq. (2)] or bending moment Pb [Eq. (1)], whichever requires the greater section.

Special Problems. Stresses in Pins.—In a pin connection between two or more structural members, the pin is designed for bending, transverse shear, and direct bearing load. A, B, C (Fig. 8) are the hinge parts of the members bearing on pin D ; find the maximum bending moment and transverse shear as usual, treating the pin as a simple beam. If the loads transmitted by A, B, C are not coplanar, resolve them into two sets of components, horizontal and vertical. Draw the moment diagram for each set, and combine the component moments at any section by the method for $P_1 P_2 P_3 - Q_1$ (Fig. 1). The maximum resultant moment is located by trial or inspection and its value inserted in Eq. (1) to find pin diameter required for bending. Component

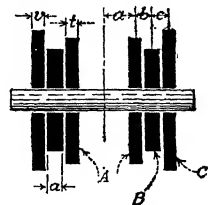


FIG. 8.—Stresses in pins.

shears are combined by the same method for each portion a , b , c of the span, and the maximum of these resultants fixes the sectional pin area required for shear, on the basis of allowable unit shear. Compute the pin diameter required by the direct bearing load of each member separately, by load $A/d.t$ = allowable unit bearing stress, etc. The maximum pin diameter by any of these computations is final.

Flat Plates in Bending under Normal Load (after Grashof).—Peripheral support, free if the periphery is unconstrained, fixed if it is riveted, bolted, or embedded; uniform load of p pounds per square inch of entire surface; maximum allowable unit fiber stress S in pounds per square inch; linear dimensions in inches; then plate thickness $t = C \sqrt{p/S}$ and maximum deflection $Y = K(p/Et^3)$. Values of C and K : For a circular plate of radius r , freely supported, $C = 0.956r$ and $K = 0.738r^4$; fixed, $C = 0.839r$ and $K = 0.176r^4$. For a square plate of side a , fixed, $C = a/2$ and $K = a^4/64$. For a rectangular plate of sides a and b , fixed, the stress being parallel to a , $C = \sqrt{\frac{b^4 a^2}{2(a^4 + b^4)}}$ and $K = \frac{a^2 C^2}{16}$. The strength of freely supported square and rectangular plates is about two-thirds that of the same plates fixed.

Closed Tank under Uniform Pressure (Thin Walls) (Fig. 9).—Uniform pressure p in pounds per square inch; linear dimensions in inches. Thickness of cylinder wall required: by tendency to rupture on diametral plane AB is

$$t = \frac{pa}{S} \quad (7)$$

by tendency to rupture on transverse plane CC is

$$t = \frac{pa}{2S}$$

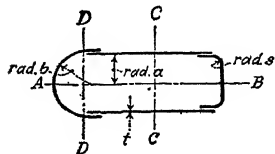


Fig. 9.—Stresses in tanks.

in each case S is the maximum allowable unit stress, compressive if p is external, tensile if internal (in which case allow for metal removed by rivet holes); if there is a riveted joint across the line of stress, multiply S by the per cent efficiency of the joint (see Structural Steel). The head A is *hemispherical*; maximum tendency to rupture occurs on diametral plane DD , and required thickness = $pb/2S$ (applicable also to a *hollow sphere*). The head B is a *dished plate*; $S = pC \frac{s}{t} + p \frac{K}{4t^2} \left[2a - s \left(1 + \frac{s}{a} \right) \right]^2$, wherein for steel heads riveted in place $C = 0.5$ and $K = 0.33$ to 0.38; for cast-iron heads cast with the cylinder $C = 0.8 = K$; solve for t (Hatt).

STRUCTURES

Framed Structures.—Conditions of static equilibrium of forces: the sum of their vertical components $\Sigma V = 0$; the sum of their horizontal components $\Sigma H = 0$; the sum of their moments about any point in their plane $\Sigma M = 0$. The structures analyzed below are *statically determinate* (or so treated); i.e., their reactions and stresses are fixed by the foregoing conditions. Analysis of a Fink roof truss under vertical load (Fig. 10a) is as follows.

Algebraic.—Equal loads (P each) on the *upper panel points*, except at ends, where the loads ($P/2$ each) are carried directly by the support, hence ignored; $R_1 = R_2 = \frac{1}{2}P$. The forces at panel point 1 (Fig. 10g) are R_1 , stress 1-2 and stress 1-3, in equilibrium; then $\Sigma V = (1-2) \sin w - R_1 = 0$, and $\Sigma H = (1-2) \cos w + (1-3) = 0$; solve for the stresses, which are compressive if plus (as 1-2) and tensile if minus (as 1-3). At panel point 2, forces P and 1-2 are now known; solve as before for 2-3 and 2-4, and at 3 solve for 3-4 and 3-5. At 4 and 5 there are three unknowns each, and a third equation is required; pass a section (wavy line) through members 6-8, 7-8, and

5-9, whose stresses hold in equilibrium all external forces to the left; with moments about 8, $\Sigma M = H(5-9) + \cos w [R_1(\text{arm } 1-8) - P(\text{arm } 2-8) - P(\text{arm } 4-8) - P(\text{arm } 6-8)] = 0$; solve for stress 5-9, which permits solution at points 5, 4, 6, and 7 by $\Sigma V = 0$, $\Sigma H = 0$. Stresses in corresponding members of the two halves of the truss are the same. For analysis by method of moments alone, see Fig. 12b and text.

Graphic.—Bow notation: letter the truss diagram Fig. 10a as shown. Then any force is known by the two letters (in clockwise order) between which it acts; i.e., stress 2-3 is JG for point 2 and GJ for point 3. In the stress diagram (b) the forces are represented in magnitude and direction by straight lines marked at their ends by the corresponding small letters. Forces represented by the sides of a closed polygon (followed around in order) fulfil the conditions $\Sigma V = 0$, $\Sigma H = 0$; any of the forces reversed is the resultant of the others. Scale the loads CD to FV on line cv ; then $tc = R_1$. For panel point 1, draw cg parallel to CG and gt parallel to GT ; their junction

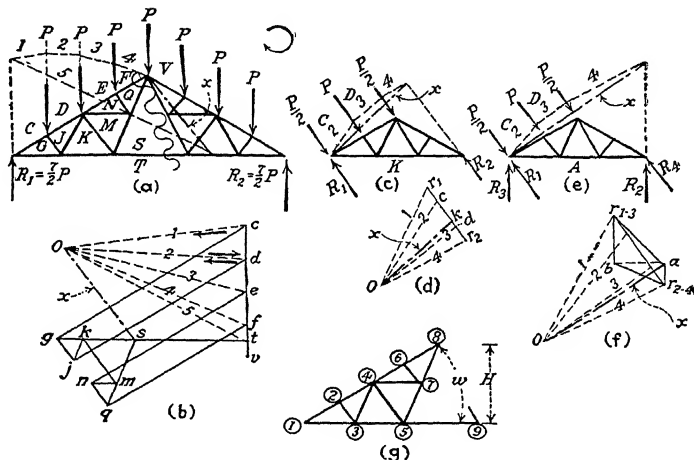


FIG. 10.—Truss diagrams.

closes the force polygon cgt and locates g ; reading clockwise, cg (compressive) acts toward point 1, gt (tensile) acts away; their lengths scale the required stresses. Similarly, j and k are located, but m is indeterminate. Pass a section through members ST , SQ , and QF , whose stresses balance R_1 , CD , DE , and EF . From any point o in (b) draw rays 1 to 5 as shown; from the peak in (a) draw string 4 parallel to ray 4 until it cuts the line of EF , thence string 3 to the line of DE , and so on until string 5 cuts the line of ST ; close the string (funicular) polygon with string x and draw ray x parallel to it in (b). Then each string cuts the two forces between which its ray terminates in (b), the string between SQ and QF being reduced to a point; triangles ocd , etc., show that each force is resolved into two forces acting along its strings in (a), the two components in each string being equal and opposite—true only if ray x locates s and defines st . Now find m , n , and q by intersection; in practice diagram (b) should be drawn for both halves of the truss.

Trusses are structures whose member stresses can be reduced to direct tension or compression only; then loads are best applied at panel points, to avoid cross-bending.

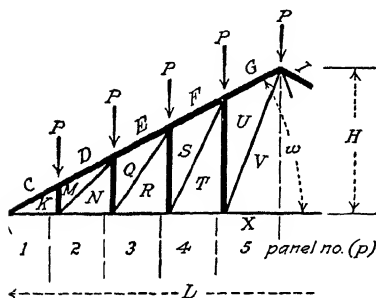
Roof trusses carry their own weight, weight of roof, and snow load as vertical loads (together with any special loads, such as cranes); wind loads are usually treated as

normal to the roof. The stresses due to vertical loading are analyzed in Figs. 10 a, b (see also Table 5). Figures 10 c, d show a truss fixed at both ends under normal wind load; scale the loads on $r_1 - r_2$. Required the values of R_1 and R_2 (whose sum ΣP is $r_1 - r_2$): draw a funicular polygon as shown (string 1 being a point because R_1 and $P/2$ are collinear); ray x , parallel to closing string x , locates k and defines $kr_1 (= R_1)$ and $r_2k (= R_2)$; solve stresses as before and combine them with the stresses due to vertical

TABLE 5.—FORMULAS FOR DEAD-LOAD STRESSES AND MEMBER LENGTHS IN TRUSSES

n = number of panels. p = ordinal number of the individual panel. $P = \Sigma P/n$
 $R_1 = \frac{P}{2}(n-1)$. The plus sign and heavy lines denote compression, the minus sign and light lines denote tension

CK identical with DM



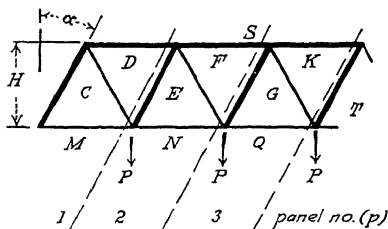
These formulas apply to corresponding members in each half-truss

PRATT ROOF TRUSS

Any number of panels

$$A = \frac{L}{H} \quad B = \sqrt{\frac{L^2}{H^2} + 4}$$

Members	Stresses	Lengths
DM, EQ, FS, GU	$+\frac{PB}{4}(n+1-p)$	$\frac{HB}{n}$
XK, XN, XR, XT, XV	$-\frac{PA}{4}(n-p)$	$\frac{L}{n}$
KM, NQ, RS, TU	$+\frac{Pp}{2}$	$\frac{2H}{n}(p-1)$
MN, QR, ST, UV	$-\frac{P}{4}\sqrt{A^2+4p^2}$	$\frac{H}{n}\sqrt{A^2+4p^2}$



WARREN BRIDGE TRUSS

Any number of panels

Members	Stresses	Lengths
Upper Chord SD, SF, etc.	$+pY \tan \alpha P$	$\frac{L}{n}$
Lower Chord CM, EN, etc.		
Web CD, DE, etc.		$H \sec \alpha$

These formulas apply to all members from one end of the truss to the other. Beginning at either end, use the plus sign for the first web member (as SC) and all members parallel thereto, and the minus sign for web members oppositely inclined; the formula itself makes the necessary reversal of sign at span center.

loads. Figures 10 e, f show a truss under normal wind load with one end on rollers (common for spans over 70 ft.); the reaction at this end is vertical; if the leeward end is on rollers, the reactions are R_1 and R_2 ; if the windward, R_3 and R_4 . Draw the funicular polygon, and extend ray x to a ; then $ar_1 = R_1$, r_2a (vertical) = R_2 , br_3 (vertical) = R_3 , $r_4b = R_4$; solve for stresses in both cases and design each member for its maximum.

Bridge Trusses.—See Table 5 for stresses due to weight of truss, which are regarded as concentrated at the panel points of the loaded chord. For live-load stresses see Moving Loads; stresses due to a series of wheel concentrations require extended treatment (see Bibliography).

Transverse Bent.—When a roof truss is carried on posts instead of walls (Fig. 11), the resulting bent is stiffened by knee braces 3-7 and 4-8. Bending of the posts due to truss deformation is ignored, and vertical loads are treated as in Figs. 10 *a*, *b*; the truss reactions become column loads on the posts. Any other loading subjects the posts to bending. Figure 11 shows horizontal wind loads, of which those on the roof are horizontal components of the normal loads, the vertical being treated as in Figs. 10 *a*, *b*. Assume the posts to be pin ended at 1 and 2 (*i.e.*, simple beams) and $H_1 = H_2 = \frac{1}{2}\Sigma P$; solve for V_1 by moments of external forces about 2; $V_2 = -V_1$. Bending loads on the posts: $Q_1 = H_2$, $Q_2 = \frac{h}{m} H_2$, $Q_3 = \frac{n}{m} H_2$; $Q_4 = (H_2 - P_5)$, $Q_5 = \frac{h}{m} (H_2 - P_5)$, $Q_6 = \frac{n}{m} (H_2 - P_5)$; maximum moment in 2-6 is $Q_1 n$ (at 4), in 1-5 is $Q_4 n$ (at 3); shears are Q_1 in 2-4, Q_3 in 4-6, Q_4 in 1-3, and Q_6 in 3-5. Direct stress: V_2 in 2-4, $+(Q_2 \operatorname{cosec} v)$ in 4-8, $(V_2 - Q_2 \cot v)$ in 4-6, $-V_1$ in 1-3, $-[(Q_5 - P_4) \operatorname{cosec} v]$ in 3-7, $-[V_1 - (Q_5 - P_4) \cot v]$ in 3-5; solve for stresses in the truss as usual. Since ΣP may act in either direction, design both posts for combined bending and direct stress [Eq. (4)] under maximum post conditions.

Buildings (Ketchum's Specifications).—Dead weight of roof truss (for spans up to 150 ft.), $W = \frac{P'}{45} \left(1 + \frac{L}{5\sqrt{C}} \right)$, where P' is the capacity of the truss, both W and P' are in pounds per square foot of horizontal projection, L is the span in feet, C the spacing of trusses in feet. For weight of roofing, see Table 6. Snow load in pounds per square foot of horizontal projection = (latitude - 30°) $\frac{A-1}{2}$, wherein $A = L/H = \text{span/rise}$ (see Table 5); for the Pacific Coast and arid regions

TABLE 6.—WEIGHT OF ROOFING (*Carnegie*)

	Pounds per square foot, inclined
Corrugated galvanized iron:	
No. 20 B.w.g.....	2.25
No. 26 B.w.g.....	1.25
Felt and asphalt or coal tar.....	2
Glass $\frac{1}{8}$ in. thick.....	1.75
Lath and plaster ceiling.....	6-8
Sheathing:	
1 in. thick, hemlock.....	2
1 in. thick, white pine, spruce.....	2.25-2.50
1 in. thick, yellow pine.....	3.50
Shingles, 6 by 18 in., 6 in. to weather	2
Slate:	
3-in. double lap, $\frac{1}{8}$ in. thick.....	4.50
3-in. double lap, $\frac{3}{16}$ in. thick.....	6.75
Zinc, No. 20 B.w.g.....	1.50

reduce by one-half; above latitude 34° the minimum load is 10 lb. per sq. ft. Wind load: pressure in pounds per square foot normal to roof (Duchemin) $= P \frac{2 \sin w}{1 + \dots}$ wherein $P = 30$ lb. per sq. ft. (in exposed locations 40) and w is as shown in Fig. 10g; pressure on vertical walls = 20 lb. per exposed square foot for heights at eaves of 30 ft. or less, 30 lb. for a height of 60 ft., and pro rata for heights between; as an example

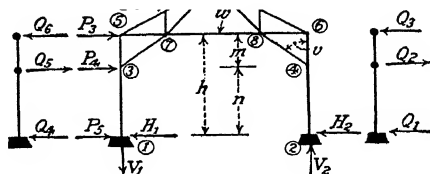


FIG. 11.—Transverse bent.

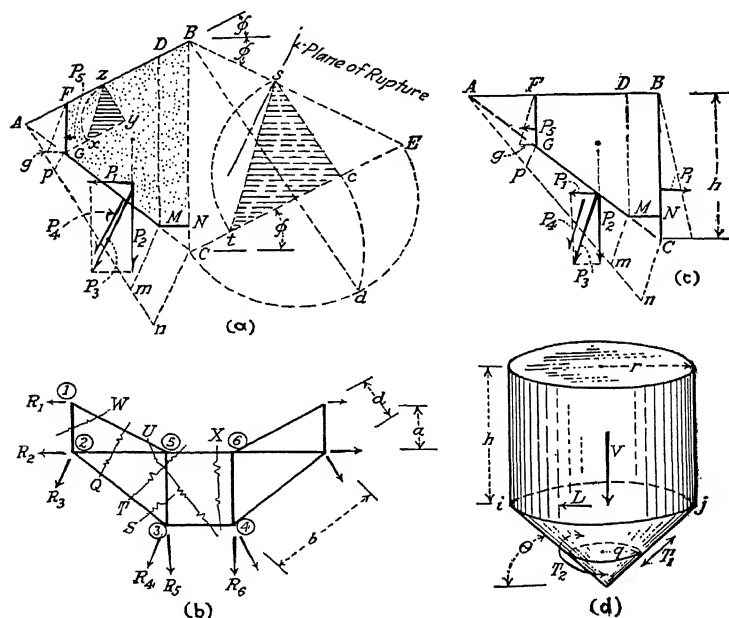


FIG. 12.—Bin loads.

of concentrating at a panel point, P_4 (Fig. 11) = unit pressure $\times \left(\frac{m}{2} + \frac{n}{2}\right) \times$ spacing of bents. Design for (1) dead and wind load, (2) dead and snow load, (3) dead, wind, and minimum snow load—whichever produces maximum stresses and stress reversals. Equivalent roof load should be not less than 30 lb. per sq. ft. of horizontal projection.

Bins.—The lateral thrust on a bin wall depends on unit weight, depth and internal friction of filling, its friction with the wall, and on size and slope of wall. Internal

friction of the filling is measured by its *angle of rest* ϕ (maximum slope at which it will not slide on itself), its friction with the wall by the maximum slope at which it will not slide on the wall material; the tangent on the latter angle is the coefficient f of this friction. Both ϕ and f vary with the nature of the filling, its fineness, and its moisture content (see Table 7, compiled chiefly from Ketchum).

TABLE 7.—CONSTANTS FOR BIN FILLINGS

	Unit weight per cubic foot, w	Angle of rest ϕ , degrees	Angle of friction with wall (whose tangent is f), degrees		
			Steel plate	Wood cribbed	Con- crete
Bituminous coal.....	47-56	35	18	35	35
Anthracite coal.....	52-56	27	16	25	27
Coke.....	23-32	..	25	40	40
Ashes.....	40-45	40	31	40	40
Ore.....	110-150	35			

A bin is *shallow* if the "plane of rupture" (see Fig. 12a) cuts the surface of fill, *deep* if it cuts the wall. The thrust diagrams (Fig. 12) show bin sections of unit length (normal to paper); then areas are represented by lines, volumes (and weights) by areas; V is the vertical, L the lateral pressure of fill in pounds per square foot, and $K = L/V$. The treatment follows Ketchum.

Shallow Bins. Example 1. Hopper bin heaped at angle ϕ ; $f = 0$; graphic method. Figure 12a, $FGMN$ is the half bin; complete $ABEC$ (AC is the hopper slope, BE and CE planes of rest). Draw successively semicircle CdE , Bd normal to CE , $Cc = Cd$, cs normal to CE and $ct = cs$; weight of area $cst = P_1$ = lateral thrust on BC acting at $BC/3$ above C . P_2 is the weight of area ABC acting through its centroid, and P_3 is the resultant of P_1 and P_2 ; draw $Cn = 2(P_3/AC)$; then area ACn represents P_3 and any intercept Mm represents unit pressure at M . Total pressure P_4 on GM is represented by area $GMmp$ acting through its centroid; pressure on MN is the weight of area $MNBD$; pressure P_5 on FG is the weight of area xyz (built on the plane of rest through G as shown), acting $FG/3$ above G ; if $Gg = 2(P_5/FG)$, intercepts to Fg are the corresponding unit pressures. Figure 12b shows the supporting bent: loads R_1 and R_2 are the reactions of a simple beam in span a due to P_5 , R_3 and R_4 those in span b due to P_4 ; $R_5 = R_6$ = pressure on MN ; multiply in each case by the spacing between bents. Cut members 1-5, 2-5, and 2-3 (section Q); then their stresses hold in equilibrium all forces to the left: ΣM (about point 2) = $-(1-5)d - R_1$, $a = 0$ and stress 1-5 = $-R_1(a/d)$, also ΣM about 5 solves for 2-3. Section S , ΣM about 2 solves for 3-5; with ΣM about 3, section T solves for 5-6 and U for 2-5; with ΣM about 5, X solves for 3-4 and W for 1-2. These are the live-load stresses in the frame; the bin walls are designed for the pressures found in Fig. 12a.

Example 2.—Hopper bin filled level; $f = 0$. Figure 12c, V (average) $w \frac{h}{2}$ and L (average) $= Kw \frac{h}{2}$; then $P_1 = Kw \frac{h}{2}$, wherein K by theory is $\frac{1 - \sin \phi}{1 + \sin \phi}$

$Kw \frac{(FG)^2}{2}$; P_2 , P_3 , and P_4 are found as before (same notation). In a triangular bin (ABC) the pressures are P_1 on BC and P_3 on AC . In a triangular bin with back wall ($FGCB$) the pressures are P_1 on BC , weight of area $Gcnp$ (acting through its centroid) on GC , and P_5 .

Deep bins of circular, square, or rectangular section, filled level (Jansen). V and L are maximum at section i ; (Fig. 12d); let H = hydraulic radius of the bin at i ; $= \left(\frac{\text{area}}{\text{perimeter}} \right)$ in feet. Maximum $V = \frac{wH}{Kf} (1 - e^{-\frac{Kfh}{H}})$, wherein W = unit weight of fill in pounds per cubic foot and e is the Napierian logarithmic base = 2.718. Maxi-

mum $L = KV = \frac{wH}{f} (1 - e^{-\frac{Kfh}{H}})$. The constant K should be determined by test; less exactly, $K = \frac{1 - \sin \phi}{1 + \sin \phi}$. Figure 12d also shows a conical bin bottom; stresses $T_1 (= \frac{1}{2} Vq \operatorname{cosec} \theta)$ and $T_2 (= Lq)$ in the cone wall are tangential. Both are maximum at $ij (q = r)$. The circular girder on which a bin of this shape is often carried falls in the class of continuous beams, which only an experienced designer should attempt; the support can be designed for simple beams, which are easier to fabricate in local shops.

Retaining walls, if vertical, are treated like bin wall FG in Fig. 12a (surcharged) or Fig. 12c (level fill). A masonry wall resists lateral thrust of fill by its weight (acting through its centroid of section); the resultant of thrust and weight of wall should fall well inside the outer edge of the wall. Roughly, the base width of a wall should be about three-sevenths of its height for level fill and four-sevenths for surcharge. A superimposed load on level fill is treated as an equivalent surcharge.

The position at which the resultant of thrust and weight is allowed to fall must take into account the bearing value of the ground.

Tanks.—For liquids $L = V = wh$ (maximum); notation as for bins.

Foundations.—For safe loads on soils, see Table 8 (Ketchum). For increasing the bearing strength of soils by piling, see Structural Timber. The line of action of the load on a foundation should fall within the middle third of the foundation base width, else the design will imply a tensile cohesion between soil and foundation that does not exist. The load is not always vertical (*cf.* Fig. 11).

TABLE 8.—FOUNDATION LOADS

	MAXIMUM SAFE LOAD, TONS PER SQUARE FOOT
Ordinary clay and dry sand mixed with clay.....	2
Dry sand and dry clay.....	3
Hard clay and firm, coarse sand.....	4
Firm, coarse sand and gravel.....	5
Shale rock.....	8
Hard rock.....	20

Impact—Working Stress.—Since the vibratory and shock effects (*impact*) of moving loads add to the stresses induced by the same loads considered as static, the allowable unit stress for “live” load is less than that for purely static or “dead” load (such as weight of structure); or, conversely, the computed stresses due to live load are increased in stated proportion before combination with dead-load stress; *i.e.*, if L and D (in pounds) are live- and dead-load stresses, respectively, as computed by static theory, then total stress (for which the member is designed) is $NL + D$. For bridges

and structures of equivalent heavy duty, N is usually taken at 2. For "structures carrying traveling machinery, such as cranes, conveyors, etc." $N = 1.25$ (American Bridge Co.). For bins, Ketchum requires that $N = 1.5$; for machinery and moving loads in headframes, tipples, and the like, $N = 2$. For the combination of wind-load stress with dead-load stress and twice the live-load stress, he permits an allowable unit stress exceeding the allowable dead-load unit stress by 25 per cent maximum. For steel buildings exposed to corrosive gases, he adds 25 per cent to the dead loads themselves before computing dead-load stresses. *All values of working stress specified in this chapter are dead-load values.*

Members subject to alternate tension and compression are designed: (Ketchum) for each kind of stress; (American Bridge Co.) for the stress requiring the greater section, their connections for the sum of the stresses. For structures, this treatment of alternate stresses is adequate, but long-continued alternations, or repetitions of one stress alone, are subject to special limitations (see Prof. Kommers' chapter in Johnson's "Materials of Construction," fifth edition).

MATERIALS OF ENGINEERING

This subdivision deals with engineering materials apart from their distinctively metallurgical use. Their choice is governed not only by strength, but also by many properties affecting either specific use or fabrication.

PHYSICAL PROPERTIES

Plasticity is measured by the extent of permanent deformation (beyond yield point) possible before failure; strictly speaking, *ductility* implies tensile, *malleability* compressive deformation of this kind, although "ductility" often denotes plastic properties in general. Plastic deformation of metals at ordinary temperatures (*cold working*) usually increases strength and hardness, but decreases ductility. Heating to a temperature just sufficient to permit molecular flow and relief of internal stress (*annealing*) reverses these effects. The work done in loading to rupture is much greater for a ductile than for a brittle material of the same ultimate strength; hence the value of ductility in resisting shock and precluding sudden failure. Ductility is defined by *elongation* and *reduction of area* (cross-sectional) accompanying failure in tension. Elongation is the percentage increase in a measured length that spans the ruptured section and, hence, combines the general stretch of the piece with local extension due to necking. The one depends on, the other is independent of, the original length selected, which is, therefore, standardized (at either 2 or 8 in.) for comparable results. A ductile bar in tension beyond its elastic limit first yields by incipient necking, thus intensifying stress in the contracted area of the neck, where rupture must ensue unless the cold-working effect of neck deformation causes a compensating local increase in strength. If this occurs, necking begins elsewhere, is again checked, and so on until the load becomes too great for compensation. The bar then fails with marked general elongation as well as reduction of area (Jeffries and Archer). A ductile material little subject to the cold-working effect fails with marked reduction of area but little general elongation, hence both criteria are required to define plastic behavior.

Bend tests form a good practical index of plasticity. They require that a test specimen of thickness t withstand bending through a specified arc around a pin of diameter d without cracking. In the specifications hereafter quoted, the notation $d = 0$ for an arc of 180 deg. indicates that the specimen must withstand bending flat on itself.

Toughness, though it implies plasticity, is marked by high resistance to plastic deformation. It is measured by work done, or energy absorbed, in loading to rupture.

Toughness under slowly applied load is distinct from toughness under impact, and the two properties do not necessarily exist together. The former is often estimated roughly by reduction of area, the latter requires an impact test. An abrupt change in the cross section of a loaded member—a shoulder, notch, or groove, even a tool scratch—induces uneven distribution of stress across the section and may lead to local overstressing at moderate load; this effect renders notched bars particularly liable to failure under impact, unless the material has “notch toughness” (Hoyt). There are various methods (Charpy, Izod, etc.) of measuring the energy of impact necessary to rupture; their results are of comparative value, but are not used as bases of commercial specification.

Repeated and Reversed Stresses.—Rapid reversals of stress (e.g., between equal tensile and compressive values) cause brittle fracture even in a ductile material at maximum intensities below static ultimate strength. In many cases, however, the material is characterized by a limiting stress intensity (*endurance limit*) within which it withstands an indefinite number of reversals. The range through which stress can oscillate indefinitely without failure is the *endurance range*; for reversal between equal intensities of opposite kind it is, obviously, twice the endurance limit. The effect of reversal between unequal maxima, or repetition of only one kind of stress, is less understood, but McAdam¹ finds that within the elastic limit of steel its endurance range remains nearly constant; i.e., that as one limit of the range is reduced the other may be correspondingly extended. Since rapid oscillation of stress occurs chiefly in machine parts, whose useful strength lies within their elastic limits, this conclusion generalizes for steels the endurance limits found for symmetrical reversal.

Hardness is variously defined as resistance to indentation, abrasion, and cutting; in most materials these resistances are unrelated. The Brinell and scleroscope tests (the only criteria of hardness commonly used) measure resistance to indentation only; the first indents the specimen by the pressure of a hardened steel ball under standard load and uses the ratio of load (in kilos) to area of indented surface (in square millimeters) as an index of hardness; the second, by means of an arbitrary scale, measures the rebound of a diamond-pointed hammer dropped on the specimen from a fixed height, the scale reading constituting the index.

Specification.—Ultimate strength, elastic and plastic properties, hardness, and physical and chemical character are general functions of the material, and form the basis of specification; whereas allowable service conditions and working stresses are partly functions of use, and govern specific design. Excerpts from standard specifications in the following pages are presented for their descriptive value and as guides to selection only; they are not sufficient to serve as forms for purchase contracts.

Specifications of the American Society for Testing Materials (A.S.T.M.) are standard expressions of quality. They are obtainable from the Society (1315 Spruce St., Philadelphia) singly or collected in triennial volumes (1924, etc.). Each specification bears a distinctive letter and number, quoted with its excerpt below.

IRON AND STEEL

Iron.—Unit weight, 0.2833 lb. per cu. in.; coefficient of linear expansion at ordinary temperatures, 0.000006 to 0.000007 per 1°F.

Constitution.—Iron, which is in the alpha state below 760°C., has at least one allotropic modification (gamma iron, nonmagnetic) occurring at temperatures above 900°C. The elemental metal (*ferrite*) is soft and ductile, its carbide (*cementite*, usually considered Fe₃C) hard and brittle; their interrelations dominate the iron-carbon series. They form solid solutions up to a limiting composition (*austenite*) of 1.7 per cent carbon at 1130°C.; at this temperature an increase in carbon content produces austenite-

. A.S.T.M., Vol. 23, p. 56.

cementite alloys, with a eutectic at 4.3 per cent carbon and an upper limit of 6.67 per cent carbon (pure cementite). Cementite is insoluble in alpha iron, whence on slow cooling austenite breaks up into cementite and ferrite, which form a eutectic (*pearlite*, called "eutectoid" for distinction) of 0.9 per cent carbon at 690°C.; below this temperature the slowly cooled alloy consists of pearlite with excess ferrite if its carbon content is less than 0.9 per cent and excess cementite if more. For relationships near the low-carbon end of the series, see Fig. 13. Rapid cooling prevents the austenite-pearlite transition or arrests it at one of its intermediate stages—martensite, troostite, or sorbite (in the order named), of which the first is notable for its hardness; this is the basis of the hardening property in steel. With 1 to 3 per cent of silicon present, the carbon requirement of the austenite-cementite eutectic is lowered, and slowly cooled primary cementite tends to break up into ferrite and graphite (as in gray cast iron), though the cementite resulting from the dissolution of austenite is less affected. Rapid cooling preserves cementite from dissociation (as in white cast iron).

Grain growth in the metal is promoted by high temperature, and coarse grain tends to persist in cooling even though the austenite-pearlite transition. It impairs strength and ductility and is corrected by reheating just above the critical temperature range of 690 to 900°C. (*normalizing*); the original structure is effaced and a fineness of grain secured corresponding to the annealing temperature. In practice, with alloys of carbon content above 0.9 per cent, reheating to about 750°C. is found to be sufficient (see Fig. 13).

Classification and Manufacture.—The definitions are condensed from those of the Committee on Uniform Nomenclature, International Association for Testing Materials. Type and quality are often defined by mode of manufacture, hence the notes on process.

Pig iron is the product of ore reduction in the blast furnace. It usually contains silicon, manganese, sulphur, and phosphorus derived from natural association, and carbon (3.5 to 4 per cent) derived from the furnace fuel. Of these, sulphur (causing red shortness) and phosphorus (cold shortness) are injurious. *Charcoal pig iron* is a high-grade product made from good grades of ore with charcoal replacing coke as fuel.

Cast iron contains so much carbon that it is not usefully malleable at any temperature. It is essentially pig iron, usually remelted, occasionally cast direct from the blast furnace. Ordinary grades contain 2.5 to 4.5 per cent carbon and 0.5 to 3.5 per cent silicon. *Gray cast iron* is relatively soft, characterized by sheetlets of graphite (gray fracture); *white cast iron* is very hard and brittle, most or all of its carbon being combined as cementite (silvery fracture); *mottled cast iron* is an intermediate type.

Malleable iron is made by annealing white iron castings under special conditions, to precipitate combined carbon as finely divided graphite (temper carbon).

Wrought iron is malleable iron aggregated from pasty particles without subsequent fusion and containing so little carbon that it does not harden usefully when rapidly cooled. It is made from white or mottled pig iron by puddling, *i.e.*, slagging off silicon and manganese and oxidizing carbon at a temperature above the melting point of pig iron but below that of the puddled product, which emerges in pasty masses retaining

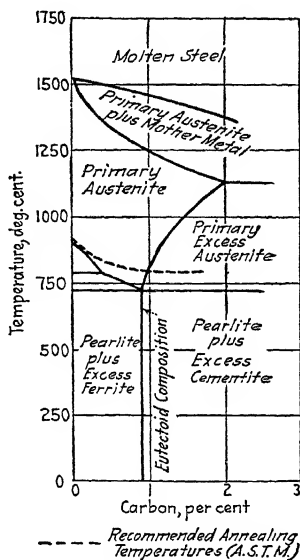


Fig. 13.—Low carbon end of Roozeboom diagram.

2 to 3 per cent of the still fluid slag; these are rolled into *muck bars*, which are finally bundled side by side and rerolled at welding temperature into *refined bar iron*, or stacked in alternate cross-layers and similarly rerolled into wrought-iron plate. Typical analysis: 0.05 to 0.25 per cent carbon; 0.10 to 0.20 per cent silicon; up to 0.10 per cent manganese; 0.02 to 0.10 per cent sulphur; 0.05 to 0.20 per cent phosphorus. Bars rolled from mixed iron and steel scrap (*common iron*) are of inferior quality. *Charcoal iron*, made in Sweden from charcoal pig and in America usually by remelting iron and steel scrap in a charcoal fire, is of exceptionally high grade.

Steel is iron usefully malleable in some one range of temperature and (1) cast from the molten state into a mass initially malleable, regardless of carbon content and hardening properties; or (2) aggregated from pasty particles without subsequent fusion, with enough carbon to harden usefully on rapid cooling from above its critical range. It is distinguished from wrought iron in point (1) by fluid origin, in point (2) by the

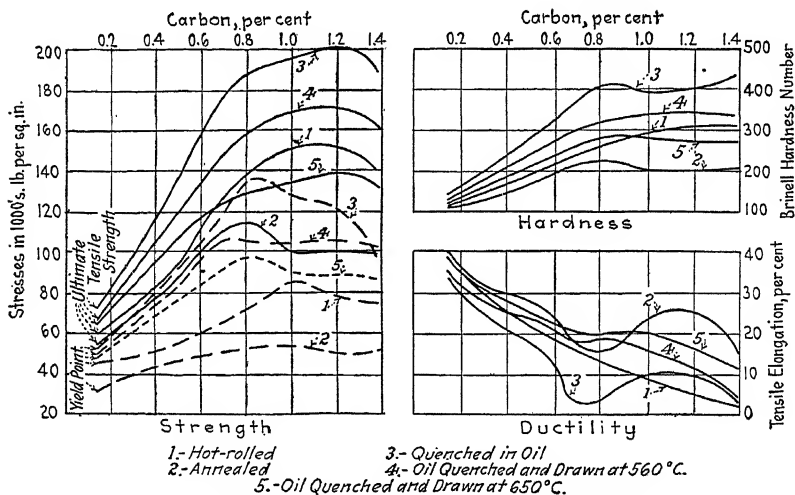


Fig. 14.—Effect of carbon content on properties of steel.

hardening property; and from malleable iron by initial malleability. *Carbon steel* owes its distinctive properties chiefly to its carbon content, *alloy steel* to one or more other constituents; *ferroalloys* contain these other constituents in excess and serve as vehicles for their introduction into iron and steel. Steel is made from pig iron by oxidizing its silicon, manganese, and carbon, reducing incidental iron oxide, and recarburizing. The Bessemer converter oxidizes by air blast, the openhearth furnace by admixture of iron oxide, each process having its "basic" practice (using basic refractories and fluxing phosphorus with lime) and "acid" practice (using siliceous linings without lime flux, hence without phosphorus removal). *Crucible steel* results from crucible melts of pure iron stock mixed with charcoal; it is of high grade, but costly. The electric furnace is used in making light castings and tool, high-speed, and special alloy steels (*e.g.*, stainless steel, certain automotive alloy steels, etc.).

Cast iron is cheap and easily cast to almost any desired shape, but is liable to defects and lacks precision of elastic properties. Silicon up to 3 to 4 per cent promotes graphitization (see Constitution) and softens the iron. Nickel has the same effect

and also refines the grain, with a compensating tendency to harden, the result in irons of high carbon content being a net increase in hardness, without impairment of machining qualities. Cast iron with 1 to 5 per cent nickel is sometimes used in electrical resistance grids, automobile cylinders and pistons, rolls, and other parts that require greater precision of performance, but ordinary gray iron is standard for structural castings, in which its disadvantages are met by high safety factors or oversize. Nonmagnetic cast iron, so alloyed that its austenitic range is shifted to ordinary temperature, is used abroad for certain electrical parts.

Gray Cast Iron.—Ordinary castings are “light” if any section is less than $\frac{1}{2}$ in. thick, “heavy” if none is less than 2 in. thick, otherwise “medium”; high-test castings include those of so-called semisteel. Table 9 shows the A.S.T.M. requirements

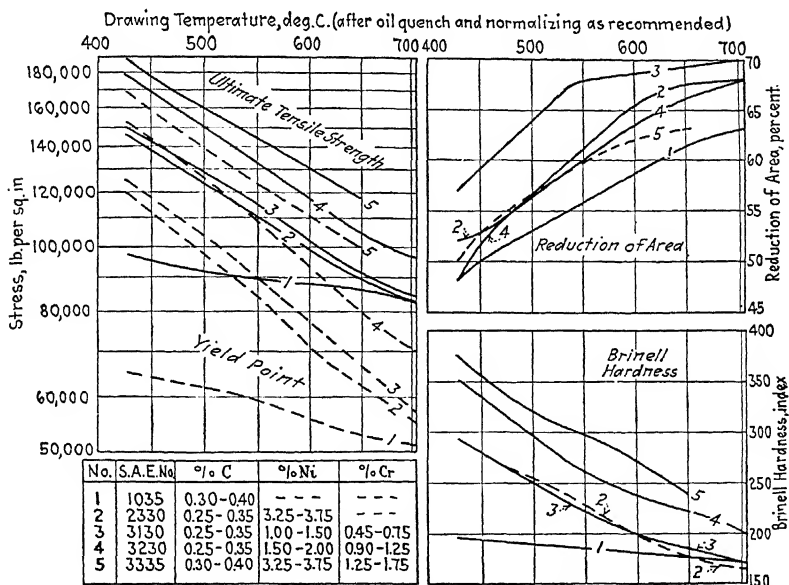


FIG. 15.—0.25 to 0.40 per cent carbon steels.

(A-48, 88), based on the “arbitration bar” (a taper cylinder 15 in. long, cast in a mold $1\frac{1}{4}$ in. in diameter at the top, $1\frac{1}{16}$ in. at the bottom). In bending, the horizontal bar is simply supported over a 12-in. span and center loaded so as to cause a central deflection of 0.1 in. in 20 to 40 sec.; the tensile test is chiefly an index of quality. Cast iron is most reliable in compression, wherein its ultimate strength is about 80,000 lb. per sq. in. (Carnegie). Its strength in shear (rarely directly employed) is about 25,000 lb. per sq. in. (Fremont). Its elastic modulus in tension and compression is 12,000,000 to 15,000,000, in shear 6,000,000 to 7,000,000 lb. per sq. in. Working stress in direct compression is 10,000 to 15,000 lb. per sq. in. Heavy impact stresses require oversize, or the use of a more shock-resistant material. For hollow cast-iron columns (New York Building law, 1917) unit load in pounds per square inch of net section = $9,000 - 40l/r_o$ (see Bending—Columns). Shrinkage stresses are reduced by designing so as to avoid the juncture of sections of contrasted thickness. Surfaces

subject to heavy wear (as wheel treads) may be hardened by *chilling*, i.e., rapid cooling in the mold to produce skins of white iron. The same effect may appear in thin sections when (due to resulting brittleness) it is not desired.

TABLE 9.—GRAY CAST IRON

Castings	Maximum per cent S	Bending, minimum at center		Minimum ultimate tensile, pounds per square inch
		Load, pounds	Deflection, inches	
Light.....	0.10	2,500	0.10	18,000
Medium.....	0.10	2,900	0.10	21,000
Heavy.....	0.12	3,300	0.10	24,000
High-test.....	3,800	0.12	28,000

Cast-iron Pipe, Bell-and-spigot (A.S.T.M., A-44).—Table 10 shows nominal inside diameter I , actual outside diameter O , and thickness T , all in inches, and weight W in pounds per foot of laid length (including socket). Class A is designed for 100-, class B for 200-, class C for 300-, class D for 400-ft. head (pressures of 43, 86, 130, and 173 lb. per sq. in. respectively). Pipes less than 20 in. in diameter must withstand a hydrostatic test of 300 lb. per sq. in.; larger sizes, 150 for class A, 200 for class B, 250 for class C, and 300 for class D. Minimum lengths, excluding socket, are 12 ft.

TABLE 10.—DIMENSIONS AND WEIGHTS OF CAST-IRON PIPE

I	Class A			Class B			Class C			Class D		
	O	T	W	O	T	W	O	T	W	O	T	W
4	4.80	0.42	20.0	4.80	0.45	21.7	5.00	0.48	23.3	5.00	0.52	25.0
6	6.90	0.44	30.8	6.90	0.48	33.3	7.10	0.51	35.8	7.10	0.55	38.3
8	9.05	0.46	42.9	9.05	0.51	47.5	9.30	0.56	52.1	9.30	0.60	55.8
10	11.10	0.50	57.1	11.10	0.57	63.8	11.40	0.62	70.8	11.40	0.68	76.7
12	13.20	0.54	72.5	13.20	0.62	82.1	13.50	0.68	91.7	13.50	0.75	100.0
14	15.30	0.57	89.6	15.30	0.66	102.5	15.65	0.74	116.7	15.65	0.82	129.2
16	17.40	0.60	108.3	17.40	0.70	125.0	17.80	0.80	143.8	17.80	0.89	158.3
18	19.50	0.64	129.2	19.50	0.75	150.0	19.92	0.87	175.0	19.92	0.96	191.7
20	21.60	0.67	150.0	21.60	0.80	175.0	22.06	0.92	208.3	22.06	1.03	229.2
24	25.80	0.76	204.2	25.80	0.89	233.3	26.32	1.04	279.2	26.32	1.16	306.7
30	31.74	0.88	291.7	32.00	1.03	333.3	32.40	1.20	400.0	32.74	1.37	450.0
36	37.96	0.99	391.7	38.30	1.15	454.2	38.70	1.36	545.8	39.16	1.58	625.0
42	44.20	1.10	512.5	44.55	1.28	591.7	45.10	1.54	716.7	45.58	1.78	825.0
48	50.50	1.26	666.7	50.80	1.42	750.0	51.40	1.71	908.3	51.98	1.96	1,050.0
54	56.66	1.35	800.0	57.10	1.55	933.3	57.80	1.90	1,141.7	58.40	2.23	1,341.7
60	62.80	1.39	916.7	63.40	1.67	1,104.2	64.20	2.00	1,341.7	64.82	2.38	1,583.3

Malleable-iron Castings.—The A.S.T.M. specifications (A-47) require a minimum ultimate tensile strength of 50,000 lb. per sq. in. and an elongation 10 per cent in 2 in.

Wrought Iron.—The hot rolling of puddled iron gives it a definite grain, along which its ultimate tensile strength is 45,000 to 55,000 lb. per sq. in., but across which it may be only 60 to 90 per cent of these values. Its tensile elastic limit is about half the ultimate. Its elastic modulus is about 27,000,000 lb. per sq. in. in tension and compression. The special value of wrought iron to the engineer lies in its welding properties.

Wrought-iron plates (A.S.T.M., A-42) are rolled from muck bars derived in turn from pig iron with permissible admixture of (1) plate scrap only (class A) or (2) wrought-iron and cast-iron scrap (class B). In Table 11 each class is subdivided into: (1) plates 6 to 24 in. wide, (2) plates over 24 to 90 in. wide; deduct 1 per cent from the tabular elongations for each $\frac{1}{16}$ -in. deficiency in specified thickness below $\frac{1}{16}$ in. Specimens for tension tests are cut with their axes parallel to the direction of rolling.

TABLE 11.—WROUGHT-IRON PLATES

	Class A		Class B	
	1	2	1	2
Ultimate tensile strength, pounds per square inch.....	49,000	48,000	48,000	47,000
Yield point, pounds per square inch.....	26,000	26,000	26,000	26,000
Elongation in 8 in., per cent.....	16	12	14	10
Cold-bend test, 90 deg. arc, $d =$	1.5 <i>t</i>	1.5 <i>t</i>	3 <i>t</i>	3 <i>t</i>
Fracture.....	All fibrous		90 % fibrous	

Steel is made by any of the processes previously described except basic Bessemer (which is foreign to American practice); it is invariably of fluid origin (definition 1). Its properties depend on composition, mechanical treatment, and heat-treatment, of which the first is universally effective, but usually associated with one or the other type of treatment in fixing the characteristics of commercial steels; the effect of carbon in these associations is shown by Fig. 14.

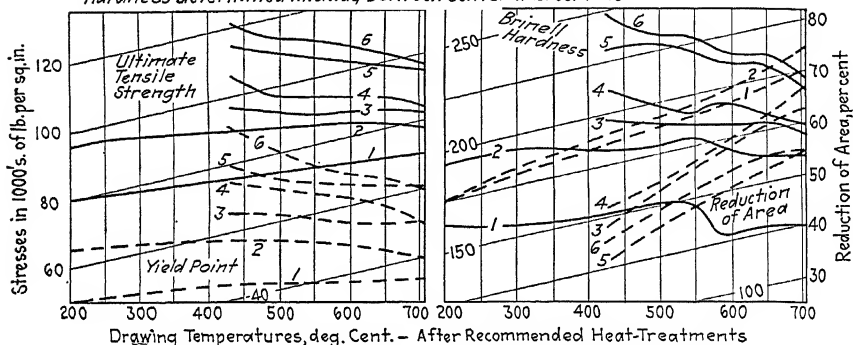
Mechanical Treatment.—Hot rolling, drawing, pressing, and forging (at temperatures above the critical range) effect a mechanical refinement of grain that is emphasized by finishing the work at temperatures not far above the critical (see Constitution); *i.e.*, hot working enlists an incidental effect of heat-treatment and increases strength and ductility. Cold working (rolling, drawing, pressing, and twisting) acts as a plastic deformation to increase elastic limit, ultimate tensile strength, and hardness, and to decrease ductility.

Heat-treatment.—Commercial steels rarely exceed 1.7 per cent in carbon content and, hence, revert to the austenitic condition when heated to or above the critical range. Complete reversion requires temperatures increasing as the carbon content varies either way from 0.9 per cent. Annealing temperatures range from 790 to 925°C. dependent on carbon content and size of piece (see Fig. 13). Subsequent cooling is effected more slowly for the higher carbon contents, and slow cooling promotes softness

TABLE 12.—MACHINE STEELS

Curve	S. A. E. No.	Quenched from, degrees Centigrade	In
1	1020	870-900	Water
2	1025	860-900	Water
3	1035	830-860	Oil
4	1035	830-860	Water
5	1045	800-830	Oil
6	1045	800-830	Water

The data on this chart apply specifically to $\frac{1}{2}$ -inch diameter or square bars. Hardness determined midway between center and surface



S. A. E. No.	Composition, per cent				Uses
	C	Mn	Max. P	Max. S	
1010	0.05-0.15	0.30-0.60	0.045	0.05	Case hardening. Cold-drawn tubing and pressed parts. Yield point (untreated or annealed), 28,000-36,000. Poor machining qualities.
1015	0.10-0.20	0.30-0.60	0.045	0.05	
1020	0.15-0.25	0.30-0.60	0.045	0.05	Case hardening. Cold-drawn tubing. Machineable.
1025	0.20-0.30	0.50-0.80	0.045	0.05	
1030	0.25-0.35	0.50-0.80	0.045	0.05	Forged, machined and cold-worked (but not case-hardened) parts.
1035	0.30-0.40	0.50-0.80	0.045	0.05	
1040	0.35-0.45	0.50-0.80	0.045	0.05	Medium-carbon steel for small and medium forgings and machined parts. Improved by normalizing 900 to 995° before quenching.
1045	0.40-0.50	0.50-0.80	0.045	0.05	
1050	0.45-0.55	0.50-0.80	0.045	0.05	Ditto for larger forgings and parts machined from bar stock. Improved by normalizing 870 to 925° before quenching.
1046	0.40-0.50	0.30-0.50	0.045	0.05	
1350	0.45-0.55	0.90-1.20	0.040	0.05	Helical springs of round cold-drawn wire. 0.025-0.100 in. in diameter (1,350) and 0.100-0.1875 (1,360).
1360	0.55-0.70	0.90-1.20	0.040	0.05	
1095	0.90-1.05	0.25-0.50	0.040	0.05	Chiefly for leaf springs.

and ductility at the expense of strength and elasticity (A.S.T.M., A-35). The range of control, from rapid to slow, is represented by (1) withdrawal from the annealing furnace and cooling in air (*normalizing*), (2) cooling in the furnace with door open, and (3) with door shut. *Hardening* involves rapid cooling (*quenching*) from above the critical range, to restrain the austenite-pearlite transition and hold the austenitic (or some intermediate) condition suspended in the cold piece. The hardening property becomes less marked as carbon content decreases, and in "ingot iron" (0.02 per cent carbon) it is absent. The usual quenching media are water, brine, and oil, of which oil is the gentlest in effect, hence the most used; a severe quench may cause warpage or even cracking. *Drawing* and *tempering* involve reheating a hardened steel to release the austenite-pearlite transition, which then proceeds to a stage determined by the drawing temperature and corresponding to the strength, hardness, and toughness desired in the finished piece (see Fig. 13). Drawing temperatures range from 200 to 700°C. *Casehardening* produces a hard skin on wrought iron or soft steel by absorption of carbon from without. The metal is packed or immersed in carbonaceous material, held at 900 to 950°C. for several hours and quenched (1) at once if toughness is immaterial, (2) after slow cooling to 650°C. and reheating to 775 to 825°C. to refine the skin, or (3) after slow cooling to 650°C. and reheating to 900 to 950°C., followed by treatment (2) and a second quenching, to refine both skin and core. Final drawing at 200°C. reduces quenching stresses (A.S.T.M., A-37). The effects of heat-treatment are more uniform in small than in large pieces (see Table 14).

Carbon Steel.—Choice of carbon content to suit required strength or projected treatment or use is indicated below by excerpts from the A.S.T.M. specifications and the "Handbook" of the Society of Automotive Engineers (S. A. E.). The latter lists a numbered series of type steels for which compositions are specified and heat-treatments and uses recommended.

Machine steels throughout the usual carbon range are described in Table 12 (S.A.E.), wherein strength properties are included only as general indications. Specified strength properties for forgings are shown in Table 14 (A.S.T.M.), which also illustrates the effect of size of piece on strength limits. For screw stock and welding steel, see Table 13.

TABLE 13.—SCREW STOCK AND WELDING STEEL (A.S.T.M., A-80)

	Per cent C	Per cent Mn	Per cent P	Per cent S
Screw stock:				
Bessemer.....	0.08–0.16	0.60–0.80	0.09–0.13	0.075–0.15
Open hearth.....	0.15–0.25	0.60–0.90	0.06 max.	0.075–0.15
Welding:				
Bessemer.....	Up to 0.12	0.60 max.	0.115 max.	0.08 max.
Open hearth dead soft.....	0.05–0.12	0.55 max.	0.05 max.	0.06 max.

The A.S.T.M. specifications (A-18) for forgings require openhearth or electric-furnace steel; phosphorus not over 0.05 per cent, sulphur not over 0.05 per cent, manganese (class A), 0.30 to 0.55 per cent, (classes B-G) 0.40 to 0.80 per cent; strength as shown in Table 14. Class A forgings are suitable for welding or casehardening, B and C are mild-steel forgings for structural use, and D, E, F, and G are machinery forgings.

TABLE 14.—CARBON-STEEL FORGINGS

Class	Size of forging, inches		Tensile strength minimum or range, pounds per square inch		Elongation in 2 in., minimum		Reduction of area, minimum		Heat treatment
	Over-all diameter or thick- ness	Wall thick- ness	Ulti- mate	Yield point	Coeffi- cient ¹	Mini- mum, per cent	Coeffi- cient ¹	Mini- mum, per cent	
A	20 max.	47,000- 60,000	0.5 ult.	1.50	2.50	..	None
B	12 max.	60,000	0.5 ult.	1.55	22	2.40	35	None
	12-20	60,000	0.5 ult.	1.48	21	2.22	32	
C	12 max.	60,000	0.5 ult.	1.70	25	2.70	38	Annealed
	12-20	60,000	0.5 ult.	1.60	24	2.52	36	
D	8 max.	75,000	0.5 ult.	1.60	18	2.20	24	None
	8-12	75,000	0.5 ult.	1.50	17	2.00	22	
	12-20	75,000	0.5 ult.	1.40	16	1.80	20	
E	8 max.	75,000	0.5 ult.	1.80	20	2.80	33	Annealed
	8-12	75,000	0.5 ult.	1.725	19	2.64	31	
	12-20	75,000	0.5 ult.	1.65	18	2.40	29	
F	8 max.	80,000	0.5 ult.	1.80	20	2.80	32	Annealed
	8-12	80,000	0.5 ult.	1.725	19	2.64	30	
	12-20	80,000	0.5 ult.	1.65	18	2.40	28	
				Elastic limit					
G	4 max.	2 max.	90,000	55,000	2.10	20.5	4.00	39	Quenched and tempered
	4-7	3.5 max.	85,000	50,000	2.00	20.5	3.80	39	
	7-10	5 max.	85,000	50,000	1.90	19.5	3.60	37	
	20 max.	5-8	82,500	48,000	1.80	19	3.40	36	

¹ Minimum elongation or area reduction (subject to tabular limit) equals (coefficient \times 1,000,000) \div ultimate strength.

Steel Castings (A.S.T.M., A-27).—Openhearth, electric-furnace, converter, or crucible steel. Class A castings are without physical requirements and are heat-treated only when so specified, or when carbon content exceeds 0.30 per cent; class B castings must meet the physical requirements of Table 15 and must be heat-treated. Treatment in either case may be by annealing, normalizing, normalizing followed by an anneal, or normalizing followed by a draw, but quenching in liquid media is excluded.

TABLE 15.—STEEL CASTINGS

Class	Maximum percentage					Minimum pounds per square inch		Elongation in 2 in., per cent	Reduction of area, per cent
	C	P		S		Ulti- mate	Yield point		
		Acid	Basic						
A	0.45	0.07	0.06	B, hard	80,000	0.45 ult.	17	25
B	0.06	0.05	0.06	B, medium.....	70,000	0.45 ult.	20	30
					B, soft.....	60,000	0.45 ult.	24	35

Steel Pipe (A.S.T.M., A-53).—For welded pipe, Bessemer or openhearth steel of soft weldable quality; pipe 3 in. or less in nominal diameter may be butt-welded; larger sizes must be lap-welded; for seamless pipe, openhearth steel only. Minimum phosphorus (openhearth), lap-welded 0.06 per cent, seamless 0.05 per cent. Two-inch or smaller pipe (except double-extra strong over 1¼ in.) must withstand a 90-deg. bend around a mandrel of twelve times its (nominal) diameter without cracking. Larger pipe (except double-extra strong) must similarly withstand flattening between parallel plates (with the weld 45 deg. from the line of pressure) to the extent of 40 per cent (butt-welded) or two-thirds (lap-welded) of its outside diameter. Table 16 lists nominal inside diameter I , outside diameter O , and thickness T —all in inches; weight W , in pounds per foot (for standard pipe, with threaded ends and couplings; for extra and double-extra, with plain ends); number of threads N per inch; and hydrostatic test pressures Pb (butt-welded) or Ps (lap-welded and seamless), in pounds per square inch.

TABLE 16.—DIMENSIONS AND WEIGHTS OF STEEL PIPE

I	O	Standard					Extra strong				Double-extra strong			
		T	W	N	Pb	Ps	T	W	Pb	Ps	T	W	Pb	Ps
½	0.405	0.068	0.25	27	700	0.095	0.31	700					
¾	0.540	0.088	0.43	18	700	0.119	0.54	700					
1	0.675	0.091	0.57	18	700	0.126	0.74	700					
1½	0.840	0.109	0.85	14	700	0.147	1.09	700	0.294	1.71	700	
2	1.050	0.113	1.13	14	700	0.154	1.47	700	0.308	2.44	700	
2½	1.315	0.133	1.68	11½	700	0.179	2.17	700	0.358	3.66	700	
3	1.680	0.140	2.28	11½	700	1,000	0.191	3.00	1,500	2,500	0.382	5.21	2,200	
3½	1.900	0.145	2.73	11½	700	1,000	0.200	3.63	1,500	2,500	0.400	6.41	2,200	3,000
4	2.375	0.154	3.68	11½	700	1,000	0.218	5.02	1,500	2,500	0.436	9.03	2,200	3,000
4½	2.875	0.203	5.82	8	800	1,000	0.276	7.66	1,500	2,000	0.552	13.70	2,200	3,000
5	3.500	0.216	7.62	8	800	1,000	0.300	10.25	1,500	2,000	0.600	18.583	2,200	3,000
5½	4.000	0.226	9.20	8	...	1,000	0.318	12.51	2,000	0.636	22.85	2,500
6	4.500	0.237	10.89	8	...	1,000	0.337	14.98	2,000	0.674	27.54	2,500
6½	5.000	0.247	12.64	8	...	1,000	0.355	17.61	1,800	0.710	32.53	2,000
7	5.563	0.258	14.81	8	...	1,000	0.375	20.78	1,800	0.750	38.55	2,000
8	6.625	0.280	19.19	8	...	1,000	0.432	28.57	1,800	0.864	53.16	2,000
9	7.625	0.301	23.77	8	...	1,000	0.500	38.05	1,500	0.875	63.08	2,000
10	8.625	0.322	28.81	8	...	1,000	0.500	43.39	1,500	0.875	72.42	2,000
11	9.625	0.342	34.19	8	...	900	0.500	48.73	1,200				
12	10.750	0.365	41.13	8	...	900	0.500	54.74	1,000				
	11.750	0.375	46.25	8	...	800	0.500	60.08	1,000				
	12.750	0.375	50.71	8	...	800	0.500	65.42	1,000				

Test pressures for pipe more than 12 in. in diameter to be calculated by $P = 24,000T \div O$, wherein P = test pressure in pounds per square inch; other notation as before.

Structural steel, used hot rolled, is usually of medium-carbon type, although for bridges the greater strength and saving in weight of nickel steel may offset its higher cost. As furnace construction resembles bridge- rather than mill-building practice in the heavy duty it imposes on steel members, data on nickel steel are added in Table 18.

Carbon steel for buildings (A.S.T.M., A-9) may be Bessemer or openhearth (except rivets, and punched plates and angles more than ¾ in. thick, which must be openhearth). All bridge steel (A-7) must be openhearth (see Table 17).

Nickel steel (A.S.T.M., A-8) must be openhearth, and show a silky or very finely granular fracture. Requirements for bridgework are shown in Table 18.

TABLE 17.—STRUCTURAL CARBON STEEL FOR BUILDINGS AND BRIDGES

	Buildings		Bridges	
	Structural	Rivet	Structural	Rivet
Composition (maxima):				
Per cent phosphorus—Bessemer.....	0.10			
Acid open hearth.....	0.06	0.06	0.06	0.04
Basic open hearth.....	0.06	0.06	0.04	0.04
Per cent sulphur.....	0.045	0.05	0.045
Tensile tests (minima):				
Ultimate strength, thousands of pounds per square inch.....	55-65	46-56	55-65	46-56
Yield point ratio.....	$\frac{1}{2}$ ult.	$\frac{1}{2}$ ult.	$\frac{1}{2}$ ult.	$\frac{1}{2}$ ult.
Yield point, thousands of pounds per square inch.....	30	25	30	25
Elongation in 8 in., coefficient ¹	1.40	1.40	1.50	1.50
Elongation in 2 in., per cent.....	22	22	
Cold-bend tests (180-deg. arc):			1-bar flats	Other
Material $\frac{3}{4}$ in. thick or less, d =	0	0	t	0
Material $\frac{3}{4}$ to $1\frac{1}{4}$ in. thick, d =	t		$2t$	t
Material over $1\frac{1}{4}$ in. thick, d =	$2t$		$3t$	$2t$

¹ Elongation in per cent = coefficient \times 1,000,000 \div ultimate strength.

The tabular elongations in 8 in. are reduced 0.25 per cent (down to 18 per cent) for each $\frac{1}{2}$ in. excess over $\frac{3}{4}$ in., and 1.25 per cent for each $\frac{1}{2}$ in. deficiency below $\frac{1}{4}$ in. in specified thickness of material. Bend-test specimen for pins and rollers to be machined to 1 by $\frac{1}{2}$ in.; for cold-drawn rivets, to be normalized before testing.

TABLE 18.—STRUCTURAL NICKEL STEEL

Composition, per cent	Plates, shapes, bars	Rivet	Physical requirements	Plates, shapes, bars	Rivet
Ni, minimum.....	3.25	3.25	Tensile tests (minima):		
C, maximum.....	0.45	0.30	Ultimate strength, thousands of pounds per square inch.....	85-100	70-80
Mn, maximum.....	0.70	0.60	Yield point, thousands of pounds per square inch.....	50	45
P (acid) maximum.....	0.05	0.04	Elongation in 8 in., coefficient ¹	1.50	1.50
P (basic) maximum.....	0.04	0.03	Elongation in 2 in., per cent.....	18	
S, maximum.....	0.05	0.045	Reduction of area, per cent.....	25	40
			Cold-bend test (180-deg. arc):		
			Material $\frac{3}{4}$ in. thick or less, d = ..	t	0
			Material over $\frac{3}{4}$ in. thick, d = ..	$2t$	0

¹ Elong. in per cent = $\frac{\text{coefficient} \times 1,000,000}{\text{ultimate strength}}$

For plates, shapes, and unannealed bars, over 1 in. thick, deduct 0.25 per cent (down to 14 per cent) for each $\frac{1}{2}$ in. excess over 1 in. Bend-test specimens of cold-drawn bars are normalized before testing.

Alloy Steels.—Alloying elements modify the properties of steel (1) through the characteristics of new alloy series, (2) by shifting the temperature ranges and transitions of the iron-carbon series. In most alloy steels, the latter effect makes carbon a principal element and induces very marked response to heat-treatment, as Fig. 15 shows. Therefore, these steels are used chiefly in the treated condition, except for

the structural nickel grade (*q.v.*). The difficulties of heat-treatment for large pieces, combined with the cost of alloy elements, thus limits alloy steels largely to the field of special machine parts, except for certain corrosion- and heat-resistant uses described in the last section of this chapter. A few ferroalloys of marked thermal or magnetic properties are noted here among the steels.

Manganese steel typically contains 11 to 13.5 per cent manganese, 1.0 to 1.3 per cent carbon, 0.3 to 0.5 per cent silicon, 0.02 to 0.08 per cent phosphorus. It therefore lies in the austenitic region of Fig. 16, has practically no magnetic properties, as originally cast is very hard and brittle, cannot be softened by anneal, and is too hard for machining. In the cast form, after an annealing quench from 1050°C. in cold water, it is extremely resistant to abrasion and cutting, though easily dented (Brinell index about 190). It shows an ultimate tensile strength of 140,000 to 150,000 lb. per sq. in., but has an ill-defined and low elastic limit and practically no yield point. It fails in tension with little necking but marked general elongation (40 to 50 per cent, compared with about 40 per cent reduction of area). In compression it deforms plastically, with low strength values. Among steels it is a poor heat conductor but of high expansivity. In small patterns it requires a shrinkage allowance of about $\frac{5}{16}$ in. per ft., and, when heated for the quench, calls for care in preventing unequal expansion and cracking.

Its uses are for abraded parts of jaw and gyratory crushers, rolls, dredges, etc., for steam-shovel dippers, frogs, switches, and curve rails, and wherever cast parts are subject to severe abrasion; it is also rolled in merchant bars and sections.

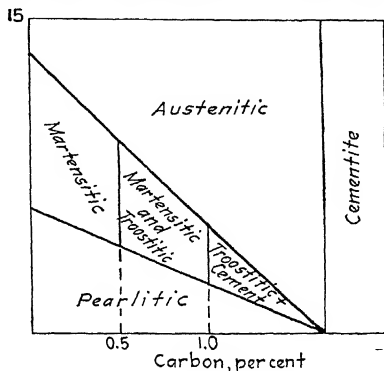


FIG. 16.—Structure of manganese steel.

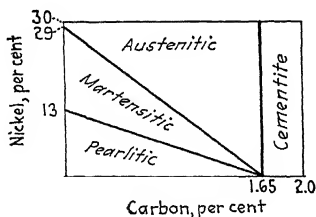


FIG. 17.

FIGS. 17-18.—Structure of nickel and chrome steel.

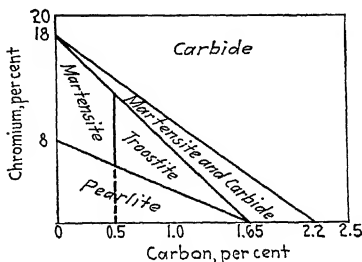


FIG. 18.

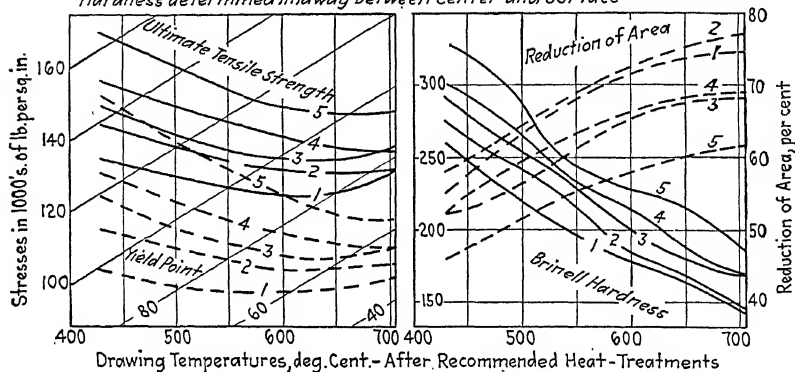
Nickel Steels.—In the pearlitic region of Fig. 17, nickel tends to refine the grain of pearlite and ferrite in the direction of sorbite and assists casehardening by retarding the rate and increasing the depth of carbon penetration; the structural and machine grades of nickel steel lie in this region (see Structural Steel). Those of the austenitic region are of limited use.

Machine grades are listed in Table 19.

TABLE 19.—NICKEL STEELS OF MACHINE GRADE

Curve	S. A. E. No.	Normalized	Quenched	
			From	In
1	2320	900–950°	800–835°	Oil
2	2320	900–950°	800–835°	Water
3	2330	900–950°	790–815°	Oil
4	2330	885–946°	790–815°	Water
5	2340	885–940°	775–800°	Oil

The data on this chart specifically apply to $\frac{1}{2}$ -inch diameter or square bars.
Hardness determined midway between center and surface



S. A. E. No.	Composition, per cent Maximum P = 0.04, maximum S = 0.045			Uses
	Ni	C	Mn	
2315	3.25–3.75	0.10–0.20	0.50–0.80	Chiefly case hardening. { Case hardening only for massive parts (core brittleness). Forgings. Machined parts. Unsuitable to case hardening. Forgings and machined parts. If machined from bar stock, normalizing may usually be omitted. Harder and more brittle after quenching, hence less used. For parts machined from bar stock, normalizing may usually be omitted. Gears of large section.
2320	3.25–3.75	0.15–0.25	0.50–0.80	
2330	3.25–3.75	0.25–0.35	0.50–0.80	
2335	3.25–3.75	0.30–0.40	0.50–0.80	
2340	3.25–3.75	0.35–0.45	0.50–0.80	
2345	3.25–3.75	0.40–0.50	0.50–0.80	Case-hardened parts with very tough core; if subsequently machined, Ni should approach lower limit.
2350	3.25–3.75	0.45–0.55	0.50–0.80	
2512	4.50–5.25	0.17 max.	0.30–0.60	

High-nickel steels (25 to 38 per cent nickel, 0.3 to 0.5 per cent carbon) are rust resistant, have low thermal expansivity, and find some use in gas-engine valves, boiler tubes, etc.

Ferronickel Alloys.—Invar (36 per cent nickel), with a thermal expansivity of 0.000001 per degree between 0 and 40°C., is used in clock construction, measuring tapes, and length standards. Its expansivity increases about 300°C. and again at low temperatures. Permalloy (about 78 per cent nickel), properly heat-treated, has extraordinary magnetic permeability, approaching saturation in the earth's field. Its magnetic qualities are impaired by mechanical strain, and therefore for cable windings heat-treatment is postponed until about the winding operation.

Chrome Steels.—In the pearlitic region of Fig. 18, chromium promotes grain refinement, strength, and hardness; this is the region of the machine steels listed in Table 20. Above the pearlitic limit there are two groups of steels noted for rust resistance; those with 9 to 16 per cent chromium and moderate carbon content can be hardened by heat-treatment; those with 17 to 30 per cent chromium can be hardened only by mechanical work and show a shock brittleness in strong contrast to the toughness of the corresponding chrome-nickel series.

TABLE 20.—CHROME STEELS OF MACHINE GRADE

S. A. E. No.	Composition, per cent					
	Cr		Max. P	Max. S		
5120	0.60-0.90	0.15-0.25	0.30-0.60	0.04	0.045	Chiefly case hardening. Physical properties those of Nos. 2320 and 3120.
5140	0.80-1.10	0.35-0.4	0.50-0.8	0.04	0.045	Heat-treated forgings and parts machined from bar stock. Equivalent to Nos. 2340 and 3140.
5150	0.80-1.10	0.45-0.55	0.50-0.80	0.04	0.045	Heat-treated forgings and parts machined from bar stock.
5210	1.20-1.50	0.95-1.10	0.20-0.50	0.03	0.030	Races and balls or rollers of anti-friction bearings.

TABLE 21.—PROPERTIES OF STAINLESS IRON

Condition	Tension, pounds per square inch		Per cent		Brinell index
	Ultimate strength	Yield point	Elongation	Reduction of area	
Hot rolled.....	164,250	77,650	11	28.2	321
Normalized, 870°C.....	81,950	51,100	26	73.4	192
Annealed, 750°C.....	73,300	42,900	35	73.6	153
Oil hardened:					
955°C, not drawn.....	173,000	157,250	12	33.7	340
Drawn 370°C.....	155,100	131,400	17	50.9	321
Drawn 540°C.....	144,650	131,900	19	65.3	293

Steel of 9 to 16 per cent chromium (Crucible Steel Co. of America), if less than 0.12 per cent in carbon, is "stainless iron"; if more, "stainless steel." When the carbon content is raised to 0.15 to 0.20 per cent, air-hardening properties develop that call for care in treatment. Table 21 lists the properties of a stainless iron (0.10 per cent car-

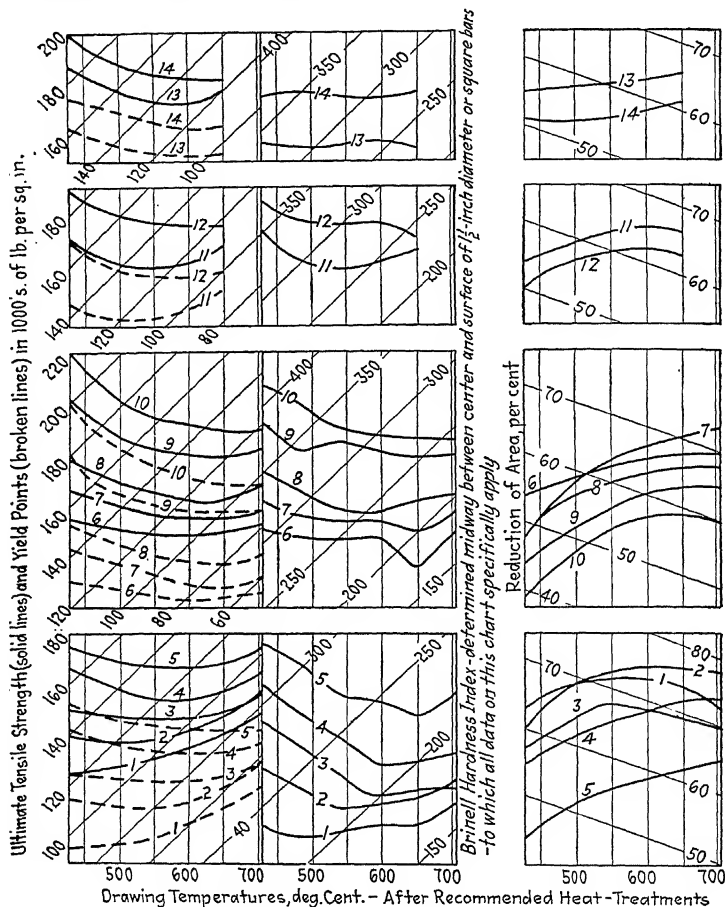


FIG. 19.—Chrome-nickel steel.

bon, 14.37 per cent chromium). Its machining qualities hot rolled are poor; normalized and annealed, good; quenched and drawn, fair.

Stainless steel, containing usually 0.30 per cent carbon and 12 to 14 per cent chromium, and hardened at about 980°C., develops its highest stainlessness and a tensile strength of 225,000 to 250,000 lb. per sq. in., both properties being retained through drawing at 480 to 510°C., after which its elongation is 7 to 9 per cent and reduction of area 20 to 25 per cent.

Chrome-nickel steels of pearlitic type furnish machinery grades of wide use. The austenitic type, recently developed, has corrosion- and heat-resistant properties described later. Its peculiar strength properties are noted here.

Machine Steels (see Table 22 and Fig. 19 in conjunction).

TABLE 22.—CHROME-NICKEL STEELS OF MACHINE GRADE

S. A. E. No.	Composition, per cent				Uses	
	Ni	Cr	C	Mn		
3115	1.00-1.50	0.45-0.75	0.10-0.20	0.30-0.60	Case hardening	(No. 3120 for massive parts only.) Both also used for heat-treated structural parts. Greater core strength; more suited to sections over 1½ in. (No. 3220 for massive parts only.) Very strong and tough core. " " " "
3120	1.00-1.50	0.45-0.75	0.15-0.25	0.30-0.60		
3215	1.50-2.00	0.90-1.25	0.10-0.20	0.30-0.60		
3220	1.50-2.00	0.90-1.25	0.15-0.25	0.30-0.60		
3312	3.25-3.75	1.25-1.75	0.17 max.	0.30-0.60		
3415	2.75-3.25	0.60-0.95	0.10-0.20	0.45-0.75	Heat treated forged and machined parts	Interchangeable with Nos. 2330 and 2335. Interchangeable with Nos. 2340 and 2345. Respond better to heat treatment and more suited to severe duty than Nos. 2330, 2335, 3125, 3130. Respond better to heat treatment and more suited to severe duty than Nos. 2340, 2345, 3135, 3140. No. 3240 not recommended for gears. Oil hardened part of very high strength for severe dynamic stress.
3125	1.00-1.50	0.45-0.75	0.2-0.30	0.50-0.80		
3130	1.00-1.50	0.45-0.75	0.25-0.35	0.50-0.80		
3135	1.00-1.50	0.45-0.75	0.30-0.40	0.50-0.80		
3140	1.00-1.50	0.45-0.75	0.35-0.45	0.50-0.80		
3230	1.50-2.00	0.90-1.25	0.25-0.35	0.30-0.60		
3325	3.25-3.75	1.25-1.75	0.20-0.30	0.30-0.60		
3435	2.75-3.25	0.60-0.95	0.30-0.40	0.45-0.75		
3240	1.50-2.00	0.90-1.25	0.35-0.45	0.30-0.60		
3335	3.25-3.75	1.25-1.75	0.30-0.40	0.30-0.60		
3340	3.25-3.75	1.25-1.75	0.35-0.45	0.30-0.60		
3450	2.75-3.25	0.60-0.95	0.45-0.5	0.45-0.5		
3245	1.50-2.00	0.90-1.25	0.40-0.50	0.30-0.60		
3250	1.50-2.00	0.90-1.25	0.45-0.55	0.30-0.60		

Maximum P, 0.04 per cent. Maximum S for the 3,100 series, 0.045 per cent; for other series, 0.04 per cent.

"*Rezistal*" (Crucible Steel Co. of America) comprises a series of austenitic steels of which the type, "Atha's 2600" contains 0.50 per cent maximum carbon, 8 per cent

Curve	SAE No.	Normalized at (°C.)	Quenched	
			From	In
1	3120	855°-885°	Oil
2	3120	855°-885°	Water
3	3130	900°-950°	815°-845°	Oil
4	3130	900°-950°	815°-845°	Water
5	3140	870°-925°	800°-830°	Oil
6	3220	885°-940°	815°-870°	Oil
7	3220	885°-940°	845°-870°	Water
8	3230	885°-940°	815°-845°	Oil
9	3240	885°-940°	800°-830°	Oil
10	3250	870°-925°	770°-800°	Oil
11	3325	970°-925°	800°-830°	Oil
12	3335	870°-925°	770°-800°	Oil
13	3435	845°-900°	770°-800°	Oil
14	3450	845°-900°	760°-790°	Oil

chromium, 22 per cent nickel, 1.75 per cent silicon, 0.70 per cent manganese, and 1 per cent copper; silicon is a requisite. They are nonmagnetic, rustless, increasingly shock resistant down to -60°C., tough at all temperatures between -185 and 1200°C., and softened by quenching. They can be hardened and strengthened only by mechanical work, whence their useful strength may be limited by machining considerations (see Table 23). Their true elastic limits are low, but in proportional limit they are comparable to other steels. They can be forged, welded, and cold-worked.

TABLE 23.—“REZISTAL” STEELS

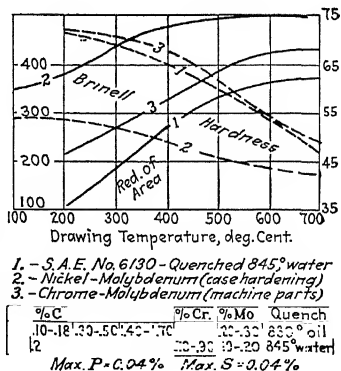
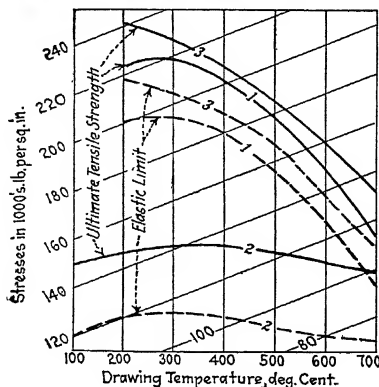
Tensile tests	Rezistal range, untreated	“Atha's 2600”—treated	
		For simple machining	For complex machining ¹
Ultimate strength, pounds per square inch	100,000–150,000	105,000–125,000	95,000–110,000
Yield point, pounds per square inch	80,000–110,000	75,000– 95,000	55,000– 65,000
Proportional limit, pounds per square inch	55,000– 70,000	35,000– 50,000
Elongation, per cent	20–35	25–35	25–35
Reduction of area, per cent	30–50	40–55	40–50
Brinell index	225–300		
Modulus of elasticity	27,000,000–30,000,000 pounds per square inch	

¹ Annealed at 870°C., water quenched.

TABLE 24.—CHROME-VANADIUM, NICKEL-MOLYBDENUM, AND CHROME-MOLYBDENUM STEELS

Chrome-vanadium steels: Cr 0.80–1.10 per cent; V 0.15 per cent minimum, 0.18 per cent desired; maximum P and S, 0.04 per cent each (except No. 6195, 0.03 per cent each).

S. A. E. No.	Other elements, per cent		Uses
	C	Mn	
6120	0.15–0.25	0.50–0.80	{ Case hardening; heat-treated structural parts. Interchangeable with Nos. 2315, 2320, 3115, 3120.
6125	0.20–0.30	0.50–0.80	
6130	0.25–0.35	0.50–0.80	{ Heat-treated forgings. Interchangeable with Nos. 2330, 2335, 3125, 3130, 3325.
6135	0.30–0.40	0.50–0.80	
6140	0.35–0.45	0.50–0.80	{ Heat-treated forgings. Interchangeable with Ni and Ni-Cr steels of similar carbon content.
6145	0.40–0.50	0.50–0.80	
6150	0.45–0.55	0.50–0.80	{ Heat-treated forgings and machined parts with more strength than is obtainable from Nos. 1045, 2345, or low-chromium steels of similar carbon content.
6195	0.90–1.05	0.20–0.45	
			Primarily a tool steel; used also in antifriction bearings.



1. — S. A. E. No. 6130 — Quenched 845° water
 2. — Nickel-Molybdenum (case hardening)
 3. — Chrome-Molybdenum (machine parts)

%C %Cr %Mo Quench
 10-18 20-50 40-70 22-32 830° oil
 20-30 10-20 845° water

Max. P = 0.04% Max. S = 0.04%

Other Alloy Steels.—Various other alloy combinations have been tried, among them the useful chrome-vanadium, nickel-molybdenum, chrome-molybdenum, chrome-nickel-molybdenum, and "silico-manganese" steels (Tables 24 and 25). Tungsten steels (Table 26) find some use, and tungsten is the chief constituent of high-speed tool steels, of which S.A.E. Nos. 71360 and 71660 are examples; these steels acquire their ability to keep a cutting edge at red heat through specialized heat-treatment and are best selected by service test.

TABLE 25.—SILICO-MANGANESE STEELS

S. A. E. No.	Composition, per cent ¹			Uses
	Si	Mn	C	
9250	1.80-2.20	0.60-0.90	0.45-0.55	Chiefly as spring steels. Also used in gears; properly treated, for cold chisels. Unsuitable to use without heat treatment.
9260	1.80-2.20	0.60-0.90	0.55-0.65	

¹ Maximum P and S, 0.045 per cent each.

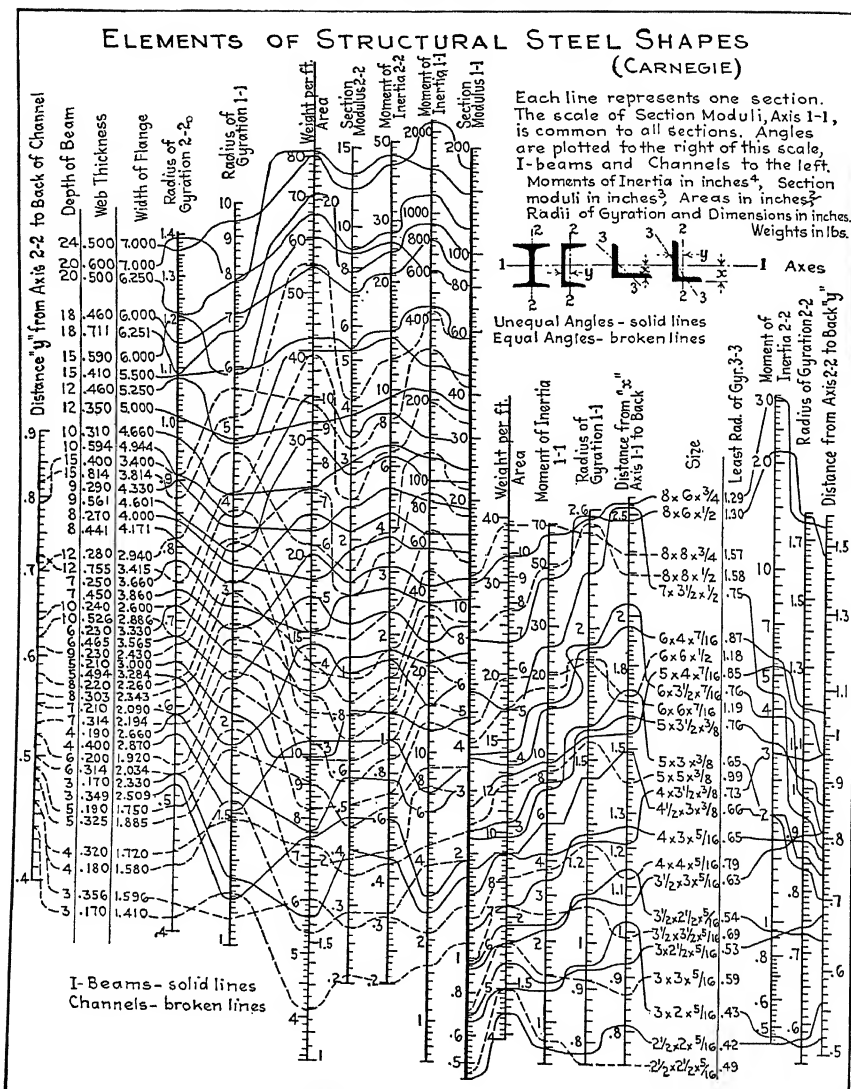
TABLE 26.—TUNGSTEN STEELS

S. A. E. No.	Composition, per cent					
	W	Cr	C	Max. Mn	Max. P	Max. S
71360	12.00-15.00	3.00-4.00	0.50-0.70	0.30	0.035	0.035
71660	15.00-18.00	3.00-4.00	0.50-0.70	0.30	0.035	0.035
7260	1.50-2.00	0.50-1.00	0.50-0.70	0.30	0.035	0.035

National Emergency Steels.—A war novelty has been the development of the so-called National Emergency steels which were intended to utilize the mixed alloy scrap coming back to the steelmakers. They have proved so useful that it will be strange if they do not last into the postwar era. In general, the 8600 and 8700 class seem to be fairly reliable substitutes for the same carbon numbers in the 2300, 3100, and 4100 steels. Much work is also being done on the 9300 and 9400 series, and it seems safe to say that any properties reported by the steelmakers can be depended upon in these numbers.

Structural-steel Design.—The selection and combination of I beam, channel, angle, tee, H beam, zee, and plate sections are based on the principle that economy requires maximum strength per pound of material with due regard to cheapness of assembly and connection, and that maximum strength per pound requires concentration of material outward from the axis of the member, with only enough connecting material to resist shear. Figure 21 shows single and built sections for various uses, and Table 27 graphically lists the elements of 69 I beams, channels, and angles, chosen for range of principal section moduli; it is a tabulating, not a calculating chart. Structural steel, especially in metallurgical use, requires frequent

TABLE 27.—STRUCTURAL SHAPES



painting, hence the objection to "boxed" members (unless concrete filled) whose interiors are inaccessible to the brush.

Riveting is designed to transmit stress by resistance of rivets to shear (not tension) and by resistance of the surrounding metal to compression. Compressive resistance depends on area td (Figs. 20 *a*, *b*), shearing resistance on area $\pi d^2/4$ in single shear (Fig. 20*a*), twice this area in double shear (Fig. 20*b*). Table 28 (after Carnegie) shows bearing and shearing values at allowable unit stresses for shop rivets of 24,000 and 12,000 lb. per sq. in., respectively; for field rivets and bolts reduce these values one-sixth (American Bridge Co.). The same table shows lengths and weights of rivets, and rules for minimum rivet spacing (including "gauge" for each section listed in Table 27). Maximum spacing (Fig. 20*i*): maximum m = eight thicknesses of piece; pitch p in line of stress of compression members (built of plates and shapes), sixteen times the least thickness of outside, or twenty times the least thickness of enclosed plate (maxima of 12 and 15 in., respectively); pitch not in line of stress, twenty-four times the least thickness riveted (American Bridge Co.). The effective area of tension members is the net after deduction of metal lost in rivet holes; diameter of hole is $\frac{1}{8}$ in. greater than that of rivet (but effective rivet diameter d is always the diameter

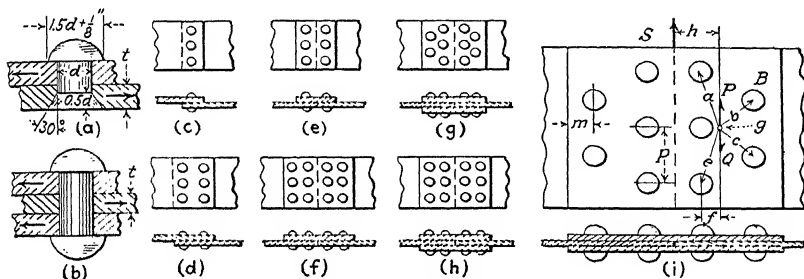


FIG. 20.- Rivets and riveted joints.

before driving). Pins are sometimes used for ease of field connection, but are giving way to modern field riveting. For steel pins, allowable unit stresses in pounds per square inch are 24,000 for bearing, 12,000 for shear, and 24,000 for extreme fibers in bending (American Bridge Co.) (see Stresses in Pins).

Riveted connections are designed with enough rivets at allowable bearing and shearing values to transmit the known stresses. In terms of strength of the unjointed plate, efficiencies of riveted joints in tension (for boilers and tanks) are as follows: for a lap joint, single riveted (Fig. 20*c*), 46 to 60 per cent; double riveted (Fig. 20*d*), 63 to 74 per cent; for a butt joint with single butt strap, single and double riveted (Figs. 20 *e*, *f*), the same; for a butt joint with double butt straps (Figs. 20 *g*, *h*), 80 per cent. Efficiencies tend to be greater for the thinner plates. In structural work, connections are often eccentric, *i.e.*, with rivets not exactly in the line of stress. Figure 20*i* shows Ketchum's treatment of a typical case. At the centroid g of the rivets (n in number) on either side of the joint, apply opposite forces P , Q each equal and parallel to stress S , which is thus replaced by force P and a couple of moment $M = Sh$. Shear per rivet due to P is S/n . In any rivet B the shear due to M is Rb , wherein R = shear due to M at unit distance from g ; resisting moment of B is Rb^2 , and total moment $M = (Ra^2 + Rb^2 + Rc^2 + Rd^2 + Re^2 + Rf^2)$; solve for R and compute the individual rivet shears. Total shear on any rivet is the sum of the shears due to P and M . The standard I-beam connections shown in Fig. 21*a* are eccentric.

Steel Members.—Axes of symmetrical members and rivet gauge lines of single or paired angles are usually identified with the center lines of truss diagrams.

TABLE 28.—RIVET CONSTANTS

Allowable Stress per Rivet—lbs.												
Shear		Value for various thicknesses of piece										
Single	Double											
2360	4710	2250	3000	3750	4500	5250	6000					
3680	7360	2810	3750	4690	5630	6560	7500	8440	9380			
5300	10600		4500			7880	9000	10130	11250			
	14430		5250		7880	9190	10500	11810	13130	14430		
for 1"	14420	18850	16000	17500	19000	20500	22000	23500	25000	26500	28000	29500

Bearing values to the left of the first heavy line are less than values for single shear; those to the right of the second, greater than values for double shear.

Rivet Spacing
Dimensions in
inches.

Minimum overlap $x = 1.5d$, or $\min x = d + \frac{1}{2}"$ (sheared edge), $d + \frac{3}{8}"$ (rolled edge)												
Rivet diam. d	$\frac{1}{2}$	$\frac{5}{8}$	$\frac{3}{4}$	$\frac{7}{8}$	1	d	$\frac{3}{8}$	$\frac{3}{4}$	$\frac{7}{8}$	1		
Minimum y	$\frac{3}{4}$	2	$2\frac{1}{16} - c$	$2\frac{5}{8}$	$2\frac{15}{16} - c$	Min. v	$\frac{7}{8}$	1	$\frac{1}{8}$	$\frac{1}{4}$	$\frac{1}{8}$	$\frac{1}{4}$
Minimum z						Std. v	$\frac{1}{8}$	$\frac{1}{4}$	$\frac{1}{8}$	$\frac{1}{4}$	$\frac{1}{8}$	$\frac{1}{4}$
Min. c for $z = 0$		$\frac{1}{2}$	$\frac{1}{16}$	$\frac{1}{8}$	$\frac{1}{4}$							
Minimum w												

Gages

Depth h	24	20	18	15	12	10	9					
Lbs. per ft.	19.9	81.4	65.4	70.0	54.7	60.8	42.9	40.8	31.8	35.0	25.4	30.0
Gage g	$\frac{7}{8}$	4	4	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{2}$	$\frac{3}{2}$	3	3	$\frac{3}{4}$	$\frac{2}{3}$	$\frac{2}{3}$
Grip k	$\frac{7}{8}$	$\frac{1}{16}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{7}{8}$	$\frac{5}{8}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Max. rivet	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$

Gages

Depth h	8	7	6	5	4	3						
Lbs. per ft.	23.0	18.4	20.0	15.3	17.25	12.5	4.75	10.3	7.7	7.5	5.7	
Gage g	$\frac{2}{16}$	$\frac{2}{16}$	$\frac{2}{16}$	$\frac{2}{16}$	2	2	$\frac{3}{16}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	
Grip k	$\frac{7}{16}$	$\frac{7}{16}$	$\frac{3}{8}$	$\frac{3}{8}$	$\frac{3}{8}$	$\frac{3}{8}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	
Max. rivet	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{5}{8}$	$\frac{5}{8}$	$\frac{5}{8}$	$\frac{5}{8}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{3}{8}$	$\frac{3}{8}$	

Gages

Depth h	15	12	10	9	8	7	6	5				
Lbs. per ft.	55.0	33.9	40.0	20.7	25.0	15.3	13.4	13.75	11.5	12.25	9.8	10.5
Gage g	$\frac{2}{16}$	2	2	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Grip k	$\frac{11}{16}$	$\frac{5}{8}$	$\frac{5}{8}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Max. rivet	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{5}{8}$	$\frac{5}{8}$	$\frac{5}{8}$

Gages

Depth h	5	4	3									
Lbs. per ft.	6.7	7.25	5.4	6.0	4.1	9.3	9.1	4.2	4	$\frac{3}{2}$	$\frac{3}{2}$	2
Gage g	$\frac{1}{16}$	$\frac{1}{16}$	$\frac{1}{16}$	$\frac{1}{8}$	$\frac{1}{8}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Grip k	$\frac{5}{16}$	$\frac{5}{16}$	$\frac{5}{16}$	$\frac{1}{4}$	$\frac{1}{4}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Max. rivet	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$

Shop Rivets
Countersunk heads $\frac{1}{4}"$ height
Heads flattened to $\frac{1}{4}"$ height
Field Rivets
Conventional Signs

near side

Beams and Girders.—A single section used as a beam is selected by Eq. (1) (Bending—Beams), which finds the moment of inertia or section modulus required by a known bending moment (see Table 27). A plate girder (Fig. 21c) is usually designed with

flange area sufficient to take full bending stresses and web plate heavy enough to take shear; required net area of angles in the tension flange = maximum $M \div Sd$, wherein M and S are as in Eq. (1) and d (effective depth) as in Fig. 21c; angles of the same size are used in the compression flange; required net web area = maximum $V \div T$, wherein V is total vertical shear and T the allowable unit shear. Part of the flange area may be supplied in cover plates, as in Fig. 21c. In all cases $S = 16,000$ lb. per sq. in., except that for a beam braced (by tie rods or otherwise) against lateral deflection at intervals greater than ten times the width (w) of the compression flange, $S = 19,000 - 300l/w$; wherein l = greatest unbraced length; l/w should not exceed

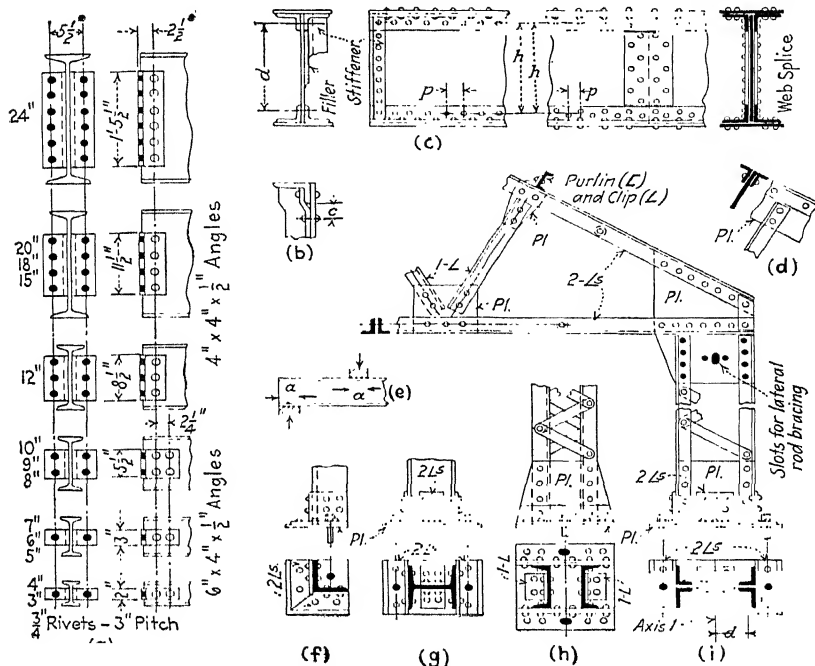


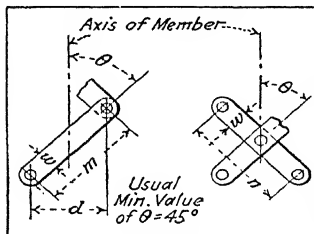
FIG. 21. Steel girders, columns, and connections.

40 (American Bridge Co.). $T = 10,000$ lb. per sq. in.; in a very short beam, loads within safe bending limits may cause excessive web shear; total shear in pounds should not exceed [depth of beam \times web thickness $\times 10,000$]. Buckling of a web (due to column action), either directly under a concentrated load or at supports, is prevented by using a web heavier than required by shear alone, or by stiffeners, or (for unsupported webs) by allowing sufficient bearing to distribute loads and reactions as follows: if d is the depth of beam, t is web thickness (both in inches), and a is as shown in Fig. 21e, then $P = St \sqrt{1 + \frac{d}{na}}$, wherein $n = 2$ for a concentrated center load and 4 for a reaction, P is the load or reaction in pounds, and $S = 19,000 - 173d/t$. The formula holds for a series of concentrated loads, provided Σa for the loads is not

heavy loads, two channels laced (Fig. 21*b*) with flanges turned either in or out combine strength in both directions with accessibility to paint. The Lally column, of steel pipe filled with concrete and threaded into cast-iron pedestal and capital, is strong, cheap, and convenient.

Table 29 lists the elements of standard I beams, whose strength about the 2-2 axis especially fits them for column use. Radii of gyration of unequal angles in

TABLE 31.—LACING BARS



Size of		w	Rivet Diam.
Channel	Angle		
15"	4", 3 1/2"	2 1/2"	7/8"
12", 10", 9"	3", 2 1/2"	2 1/4"	3/4"
8", 7", 6"	2", 2"	2"	5/8"
5"	2"	1 3/4"	1/2"

pairs are shown in Table 30, which may also be used for the angle column (Fig. 21*i*) because lacing is not computed as part of the column section. In this figure, if d is as shown, R_0 is the radius of gyration of the whole column about its 1-1 axis, and r_0 that of one pair of angles about their own 1-1 axis (Table 30), then $R_0 = \sqrt{r_0^2 + d^2}$; if a web plate is used, $R_0 = \sqrt{I \div \text{area}}$ as computed in Table 1. For lacing bars, see Table 31 (American Bridge Co.); if d exceeds 15 in., use double lacing.

Steel Buildings.—Pitch of roof (H/L , Table 5): for a tar-and-gravel roof, almost flat; for corrugated steel (Table 32), preferably one-fourth; for slate, tile, and shingles, minimum one-fourth, preferably one-third. Economical spacing of trusses usually varies from 16 (for spans up to 60 ft.) to 25 ft. for long spans and heavy loads (Ketchum); these intervals permit using single lengths of rolled sections (e.g., channels) as purlins.

Details.—Figure 21*i* shows a typical connection of column and roof truss. A good section for many purposes (as for chords of roof trusses, Fig. 21*i*) consists of two angles back to back with enough space between for plate connections. They should be riveted together, with fillers between, at intervals of 2 to 4 ft. Figure 21*d* shows a variant of this section with web plate, used for the upper chords of roof trusses on which purlins are carried between panel points. Secondary truss members may be single angles; the eccentric stress due to connecting such angle by only one leg must be taken into account. Minimum thickness of metal (except in fillers), 1/4 in.; minimum angle, 2 × 2 × 1/4 in.

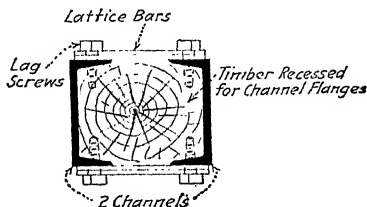


FIG. 22.—Steel and timber beam.

TABLE 32.—WEIGHTS OF FLAT AND CORRUGATED STEEL SHEETS (*Ketchum*)
½-in. corrugations)

Gage No.	Thickness, inches	Pounds per square (100 sq. ft.)			
		Flat		Corrugated	
		Black	Galvanized	Black-painted	Galvanized
16	0.0625	250	266	275	291
18	0.0500	200	216	220	236
20	0.0375	150	166	165	182
22	0.0313	125	141	138	154
24	0.0250	100	116	111	127
26	0.0188	75	91	84	99
28	0.0156	63	79	69	86

For two corrugations side lap and 6-in. end lap, add 25 per cent; for one corrugation side lap and 4-in. end lap, add 15 per cent. For paint, add 2 lb. per square.

General.—The elastic moduli of practically all steels lie between 27,000,000 and 30,000,000 lb. per sq. in.; for structural steel, 29,000,000 is a fair value. Endurance limits are still under investigation. McAdam's results,¹ summarized in Table 33, include determinations of torsional endurance limits which, though tentative, have comparative value.

NONFERROUS METALS AND ALLOYS

Nonferrous Metals and Alloys.—Campbell's comprehensive list of alloy compositions is available in pamphlet form (A.S.T.M.). Materials of value for their resistance to rust at ordinary temperatures are treated here, as being of general engineering use; heat- and chemical-resistant alloys are treated in the third section of this chapter.

Copper.—Unit weight, 0.3214 lb. per cu. in.; specific gravity, 8.89; expansivity, 0.0000168 per degree centigrade at ordinary temperatures. The resistivity of the pure metal is lower than that of any other except silver, but rises rapidly with the addition of certain impurities; expressed conversely, its conductivity is reduced 1 per cent by 0.0013 per cent arsenic or 0.0071 per cent antimony (Hofman). Copper is embrittled by bismuth, arsenic, antimony, lead, iron, and sulphur; arsenic is purposely used as a hardener and to increase strength. The American market refers to electrolytic refined shapes—wire bars, wedge bars, cakes, slabs, billets, ingots, and ingot bars, of which the last two are made for remelting, the remainder for rolling, drawing, etc. Lake copper (electro or fire refined) must originate in the northern peninsula of Michigan. Table 34 shows the A.S.T.M. requirements for both (B-5 and B-4, respectively), wherein cakes, slabs, and billets take wire-bar classification only when specified for electrical use at purchase. Casting copper (fire refined) is often specified at 99.5 per cent copper minimum, although selected brands are of much higher grade. Standard copper (in Europe) comprises blisters and rough coppers, many of them specialized in clientele. The properties of copper respond sharply to

¹ *Proc. A.S.T.M.*, Vol. 23, Part 2, p. 56.

TABLE 33.—TENSILE, TORSIONAL, AND ENDURANCE DATA FOR STEELS

Stresses, in thousands of pounds	Composition, per cent					Tension		Torsion	
	C	Mn	P	S	Si	Ultimate	Endurance limit	Ultimate	Endurance limit
Water quenched.....	0.023	0.037	0.002	0.031	0.005	46.7	25.0	40.7	
Cooled in furnace.....						55.1	22.5	46.6	
Cooled in air.....						60.7	21.5	48.7	
Oil quenched and drawn.....	0.24	0.45	0.009	0.021	0.007	60.3-68.5	27.0	52.3-60.5	
Water quenched and drawn.....						67.1-79.1	30.5	54.3-60.2	
Cooled in furnace.....						70.3	29.5	55.5	17
Cooled in air.....						77.5	32	67	
Oil quenched and drawn.....	0.35	0.57	0.023	0.053	0.04	84-91.5	33.5	50-65	17.5
Water quenched and drawn.....						82-97.3	37.0	60.5-67.5	19.5
Cooled in furnace.....						80.7	31.5	70.6	19
Cooled in air.....						114.6	39.5	73	
Cooled in furnace.....	0.51	0.31	0.019	0.020	0.13	59.7	33.5	60.9	22.5
Cooled in air.....	0.37	0.70	0.017	0.016	0.12	113.6	51	77.9	38
Oil quenched and drawn.....	0.42	(3.60 per cent Ni)				120.5	63	79	
Water quenched and drawn.....						132.2-154.2	73.5	86.9-93.3	46.7
Cooled in furnace.....						89.4	45.2	61.2	28
Cooled in air.....	0.31	0.61	0.028	0.028	0.13	91	42.5	67.3	
Oil quenched and drawn.....			(3.35 per cent Ni)			97.5-155.5	47.5-63.5	67.9-111.4	35-48
Water quenched and drawn.....						106.2-143.3	58.3-67	69.7-91.9	
Cooled in furnace.....	0.39	0.29	0.034	0.035	0.195	85.7	38.5		
Water quenched and drawn.....						121.3-136.6	65.2		
Cooled in furnace.....	0.31	0.44	0.034	0.035	0.22	82.4	36.5	62.9	
Water quenched and drawn.....						139.9-164.7	63.5-72.5	86-104	
Cooled in furnace.....	0.50	0.48	0.030	0.039	0.24	111.3	49.5	76.7	
Water quenched and drawn.....						157-172	77-88	103.3-125.5	
Cooled in furnace.....	0.55	101.7	44.5	77	
Water quenched and drawn.....						147.7-201	78.5-94.5	110.5-124	
Cooled in furnace.....	0.49	117.7	50	78.5	
Water quenched and drawn.....						144.2-164.5	72-84.5	82.4-90.3	

"Endurance limits" are maximum experimental values of stress applied for 10,000,000 cycles of reversal without causing failure. In general, the tabular ranges for quenched and tempered steels show range of values due to variations in drawing temperatures.

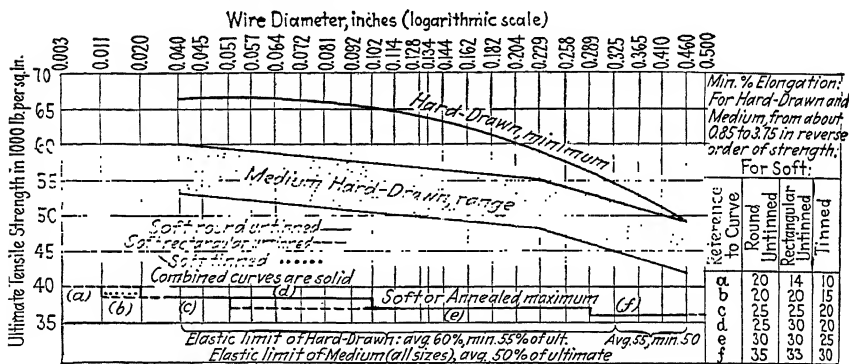
the mechanical treatment involved in fabrication and are, therefore, defined here for typical fabricated forms; heat-treatment does not extend beyond annealing.

Cuproberyllium (Be about 2.25) comes in a class by itself; in view of its high-tensile, high-fatigue limit. Its corrosion resistance is much greater than that of copper and it has no liability to sparking.

TABLE 34.—REFINED-COPPER SHAPES

	Resistivity at 20°C., ohms per meter-gram	Per cent	
		Cu + Ag	Cu + Ag + As
Electrolytic and low-resistance Lake:			
Wire bars.....	0.15535	99.880	
Ingots and ingot bars.....	0.15694	99.880	
High-resistance Lake, defined as.....	over 0.15694	99.880 ^a

^a Arsenic content subject to agreement at purchase.

TABLE 35.—COPPER WIRE (A.S.T.M.)
Maximum resistivity in pounds per mile-ohm

Diameter or thickness of wire, inches	0.460	0.324	0.289	0.102	0.020	0.011	0.003
	0.325	0.290	0.103	0.021	0.012		

Untinned:

Hard-drawn, round (B-1).....	900.77	910.15	910.15	910.15	910.15	910.15	910.15
Medium hard-drawn, round (B-2)....	15	905.44	905.44	905.44	905.44	905.44	905.44
Soft annealed, rectangular or round ¹ ..	891.58	891.58	891.58	891.58	891.58	891.58	891.58
Tinned, soft (B-33).....	896.15	896.15	900.77	910.15	929.52	939.51	

¹ B-48 (tentative) for rectangular, B-3 for round.

Sheet copper is rolled in all thicknesses up to $\frac{1}{4}$ in.; thicker sections are usually known as plate. To convert thickness (in inches) into the common designation by weight in ounces per square foot, multiply by 750.

Roofing Copper.—For flashings, gutters, and leaders of rolled sheet, 16-oz. copper is preferable; sheet of "roofing temper" (annealed) is used in flashings and other pieces shaped on the job, of "cornice temper" (hard rolled) for gutters, leaders, cornices, etc., supplied ready shaped. Provided free movement is allowed, flashings up to 12 in. in width may be fastened with copper or brass nails, but larger sizes require copper cleats (Copper & Brass Research Association). Copper shingles of rolled sheet are best suited to elaborate buildings, but the copper-clad asphalt shingle competes with slate and composition roofing for the small dwelling (Anaconda Copper Mfg. Co.).

Boiler tubes (A.S.T.M., B-13) of the compositions shown in Table 34 may be arsenical (0.15 to 0.35 per cent arsenic) or nonarsenical, finished hard or annealed. They are tested to 750 lb. per sq. in. internal hydrostatic pressure.

Painting, rarely advisable, requires preliminary roughening of the copper to secure bond. Wash the surface with a solution of 4 oz. of copper sulphate and $\frac{1}{8}$ oz. of nitric

TABLE 36.—COPPER AND BRASS PIPE

Size, inches	Outer diameter <i>O</i> , inches	Standard			Extra strong		
		Thick- ness <i>T</i> , inches	Nominal pounds per foot		Thick- ness <i>T</i> , inches	Nominal pounds per foot	
			Copper	Brass		Copper	Brass
$\frac{1}{8}$	0.405	0.0620	0.259	0.246	0.100	0.371	0.353
$\frac{1}{4}$	0.540	0.0825	0.460	0.437	0.123	0.624	0.593
$\frac{3}{8}$	0.675	0.0905	0.643	0.612	0.127	0.847	0.805
$\frac{1}{2}$	0.840	0.1075	0.957	0.911	0.149	1.25	1.19
$\frac{3}{4}$	1.050	0.1140	1.30	1.24	0.157	1.71	1.62
1	1.315	0.1265	1.83	1.74	0.182	2.51	2.39
$1\frac{1}{4}$	1.660	0.1460	2.69	2.56	0.194	3.46	3.30
$1\frac{1}{2}$	1.900	0.1500	3.20	3.04	0.203	4.19	3.99
2	2.375	0.1565	4.23	4.02	0.221	5.79	5.51
$2\frac{1}{2}$	2.875	0.1875	6.14	5.83	0.280	8.84	8.41
3	3.500	0.2190	8.75	8.31	0.304	11.82	11.24
$3\frac{1}{2}$	4.000	0.2500	11.41	10.85	0.321	14.37	13.67
4	4.500	0.2500	12.94	12.29	0.341	17.25	16.41
$4\frac{1}{2}$	5.000	0.2500	14.46	13.74	0.375	21.10	20.07
5	5.563	0.2500	16.21	15.40	0.375	23.67	22.52
6	6.625	0.2500	19.41	18.44	0.437	32.93	31.32
7	7.625	0.2815	25.17	23.92	0.500	43.34	41.23
8	8.625	0.3125	31.63	30.05	0.500	49.42	47.02
9	9.625	0.3440	38.83	36.94	0.500	55.56	52.81
10	10.750	0.3655	46.22	43.91	0.500	62.40	59.32

Hydrostatic test pressure in pounds per square inch = $2 TS \div O$, wherein *S* in pounds per square inch is 6,000 for copper, 7,000 for brass.

acid in $\frac{1}{2}$ gal. of lukewarm water; wash off the solution and apply a first coat of 15 lb. of red lead in 1 gal. of raw linseed oil with not more than $\frac{1}{2}$ pt. of oil drier; finish with two coats of 15 lb. of white lead in 1 gal. of the oil, with the desired color and not more than 5 per cent of the drier (Copper & Brass Research Association).

Brasses and Bronzes.—The useful range of copper-zinc alloys includes the malleable and ductile alpha and beta phases; that of the copper-tin alloys includes the alpha and alpha-plus-delta fields. By additions of tin the brasses are gradually merged in the tin bronzes, and many alloys of both groups contain essential elements foreign to their respective binary series. Usage has extended the term "bronze" to several brasses, as well as to certain copper-aluminum alloys; but composition and treatment affect strength and ductility so profoundly that each alloy presents an individual range of properties, to which the usual terminology is a poor guide. Cold work, which gives the brasses their strength and hardness, also induces internal stresses that sometimes act later under mild corrosive conditions to cause "season" cracking and splitting, and, if strength is a requisite, the remedy of annealing must not be carried too far. A common test for sufficiency of anneal requires immersion for 15 min. without cracking in an aqueous solution of 100 g. of HgNO_3 and 13 cc. of HNO_3 (specific gravity 1.42) per liter. For bronzes as bearing materials, see Bearing Metals.

Casting Brass.—Table 37 shows the A.S.T.M. specifications (B-30) for ingot metal suited to sand castings. No. 1 is a high-grade steam metal, No. 2 a high-grade, and No. 3 a medium-grade red brass for general service; these alloys are easily handled in the foundry and machine readily. No. 4 is suited to low-pressure valves and fittings, No. 5 to air, gas, and water fittings; neither is difficult to handle, and both machine very easily. No. 6 is a yellow brass for general service, No. 7 is used for plumbers' flanges, scuppers, etc.; both are difficult to machine and to handle in the foundry, although their casting properties are improved (but with more shrinkage) by adding up to 0.3 per cent aluminum; neither is suited to use in bearings or water-pressure fittings.

TABLE 37.—BRASS INGOT METAL

No.	Standard composition, per cent				Maximum, per cent impurities ¹			Ultimate tensile ²	Elongation in 2 in., per cent	Reduction of area, per cent	Brinell index
	Cu	Sn	Pb	Zn	Fe	Sb	P				
1	87±1	8±0.75	2 max.	3±0.50	0.25	0.25	0.05	30-36	25-30	25-30	
2	85±1	5±0.50	5±0.50	5±0.50	0.35	0.25	0.05	27-33	16-20	15-20	50-60
3	83±1	4±0.50	6±0.75	7±0.75	0.35	0.25	0.05	28-33	15-20	20-26	55-60
4	77±1	3±0.50	10±0.75	10±0.75	0.40	0.35	0.05	23-28	10-15	10-15	50-55
5	76±1	2±0.50	6±0.75	16±1.00	0.40	0.25	0.05	25-30	20-30	20-30	43-48
6	65±1	1.0 max.	2±0.50	33±1.00	0.50	0.20	0.02	25-30	20-30	25-35	37-42
7	60±1	1.5 max.	3±0.50	37±1.00	1.00	0.20	0.02	30-45	15-25	20-30	43-48

¹ Sulphur, 0.05 per cent maximum. Aluminum, zero, except as noted for Nos. 6 and 7.

² In thousands of pounds per square inch. Only the compositions are specified.

Naval brass, containing a small proportion of tin, is one of the few brasses that respond favorably to quenching and drawing, the effect of which is shown in Fig. 23. Quenching produces a mixture of the alpha and beta constituents; drawing at 300°C. or more permits the beta to break down, with sacrifice of ductility due to precipitation of the gamma. Therefore the drawing temperature should not exceed 250°C.¹ Naval

¹ CAMPBELL, WILLIAM, Report to Navy Department.

brass rod as specified by the A.S.T.M. (B-21) for machine parts, bolts, etc., can be forged hot, but is not free cutting; it is hot rolled or extruded, then if necessary cold

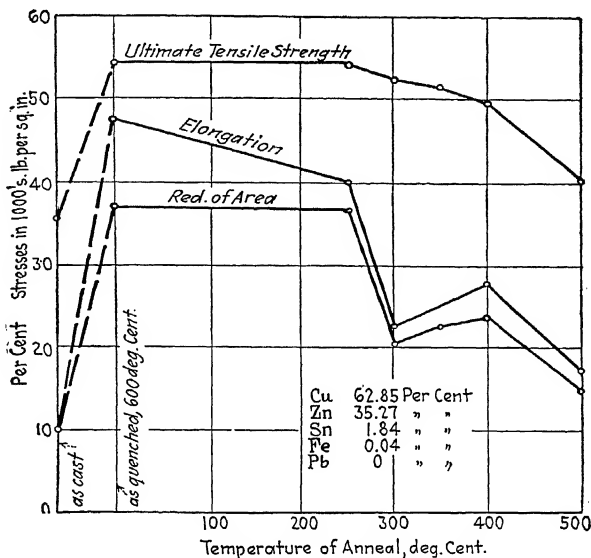


FIG. 23.—Heat treatment of Naval brass.

drawn to size (see Table 37). Specimens must stand a 120-deg. bend ($d = 2t$) and pass the $\text{Hg}(\text{NO}_3)_2$ test.

TABLE 38.—ROD BRASS

Rod	Composition (Zn remainder)				
	Per cent Cu	Per cent Pb	Per cent Sn	Maximum per cent Fe	Maximum miscellaneous
Forging.....	58.5-61.5	1.5-2.5	0.15	0.35 ^a
Free-cutting.....	60.0-63.0	2.25-3.25	0.15	0.25 ^a
Naval brass	59.0-62.0	0.30 max.	0.50-1.50	0.10	0.10
	Diameter or thickness	Pounds per square inch		Elongation in 2 in. per cent	
		Ultimate tensile strength	Yield point		
	Up to 1 in.	62,000	31,000	25	
	Over 1-2.5 in.	60,000	30,000	30	
	Over 2.5-3.5 in.	56,000	25,000	35	
Over 3.5 in.	54,000	22,000	40		

^a Including Fe

Rod Brass [Table 37 (A.S.T.M.)].—(1) Free-cutting rod (B-16) is finished by cold drawing and (as a limit to the resulting hardness) is required to stand a 120 deg. bend for which $d = 2t$. It is suited to high-speed screw machine work, and for proper machining quality must contain lead, which, however, reduces strength and ductility. (2) Forging rod (B-15) is hot rolled or extruded, then if necessary cold drawn to size; a specimen one diameter long must withstand while hot an 80 per cent compressive reduction of length without cracking. It is suited to hot forging and machines readily; its strength, forging quality, and maximum lead content (without danger of hot shortness) vary inversely, its toughness directly, with copper content.

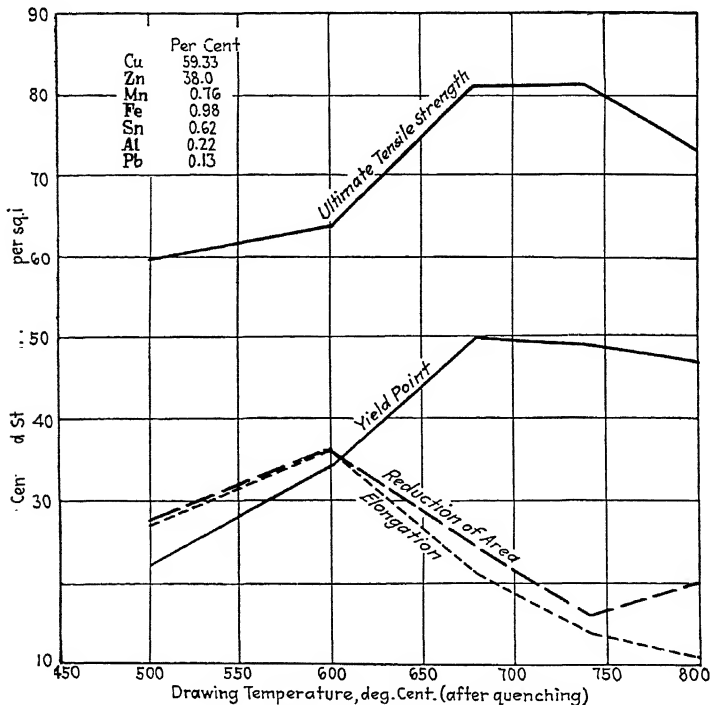


FIG. 24.—Properties of manganese bronze.

Brass pipe for plumbing, boiler feed lines, etc., (A.S.T.M., B-43) is finished by cold drawing to size and for ordinary purposes is semiannealed; if intended for bending, it must be so specified and is then fully annealed. For sizes and weights see Table 36; the HgNO_3 test and various flattening tests are required.

Tobin bronze (American Brass Co.) is very similar in composition to Naval brass and meets the same strength requirements. It is one of the nonferrous alloys available in structural shapes (I beams, bulb angles, tees) and is also marketed in bars, hot-forged and pressed parts, sheets, plates, tubes, rods, and wire. It resists the action of sea water and plays a part in ship construction. Unit weight, 0.3036 lb. per cu. in.; specific gravity 8.404.

Muntz metal comprises brasses of the 57 to 63 per cent copper range. 70-30 brass defines itself. *Admiralty metal* is typically a 70-29 alloy with 1 per cent tin. For seamless condenser tubes, these alloys as specified by the A.S.T.M. are listed in Table 39; they are cold-drawn to size and sufficiently annealed, with Muntz metal, to show a microstructure indicating complete recrystallization, and with 70-30 and Admiralty metal, to produce an average grain size of not more than 0.045 mm.

TABLE 39.—CONDENSER TUBES

A.S.T.M.	Composition, per cent ¹	Cu	Sn	Maximum Pb	Maximum Fe
B-55	70-30 brass.....	Min. 70	0.075	0.06
B-56	Muntz metal.....	59-63	0.30	0.07
B-44	Admiralty metal.....	Min. 70	0.9-1.2	0.075	0.06

¹ Remainder Zn. Internal hydrostatic test pressure, 1,000 lb. per sq. in.

TABLE 40.—CAST MANGANESE BRONZE (A.S.T.M.)

Per cent Cu	Per cent Zn	Per cent Sn	Per cent Mn	Per cent Al	Per cent Fe	Per cent Pb
55-60	38-42	0-1.5	0-3.5	0-1.5	0-2	0-0.2

TABLE 41.—CAST MANGANESE BRONZE (*Moore*)

Pounds per square inch	Tension	Compression	Shear ¹	Torsion	Composition, per cent
Ultimate strength.....	70,000	42,390	43,190	Cu 56.85
Proportional limit.....	13,000	15,500	11,140	Zn diff.
Elastic limit.....	10,240	Sn 0.32
Elastic modulus.....	14,235,000	Mn 0.20
Modulus of rupture.....	61,890	Al 0.23
Endurance limit.....	15,000	Fe 1.50

Tensile elongation in 2 in., 32.8 per cent; reduction of area, 40.8 per cent.; Brinell index, 93, specific gravity, 8.26.

¹ Double shear.

Manganese bronze comprises a group of brasses of varying composition but with strength properties comparable to those of steel, due to minor percentages of added elements. Their response to heat-treatment varies considerably. Figure 24 shows the effect of quenching and drawing on one member of the group, but many others if quenched lose their ductility beyond recovery.¹ Cast manganese bronze (A.S.T.M., B-54, tentative) must show an ultimate tensile of at least 65,000 lb. per sq. in. and

¹ CAMPBELL, WILLIAM, Report to Navy Department.

TABLE 42.—SHEET BRASS

Composition, per cent	High sheet	Cartridge brass	Temper	Minimum ultimate tensile strength, pounds per square inch	Mini- mum per cent elon- gation ¹	Brinell index ²
Copper.....	64.5-67.5	66.5-69.5	High sheet (hard): Quarter hard..... Half hard..... Hard..... Extra hard..... Spring..... Same (annealed): Light anneal..... Drawing anneal.... Soft drawing anneal Cartridge brass.....			
Maximum lead.....	0.30	0.07		45,000	27.5	75- 95
Maximum iron.....	0.05	0.05		52,500	15.0	95-115
Maximum miscella- neous.....	0.10	0.15		67,500	5.0	130-150
Zinc.....	Remainder			80,000	2.0	150-170
				87,500	1.0	160-180
				45,000	32.0	65- 75
				42,000	38.0	55- 65
				40,000	42.0	47- 55
				51- 65

¹ For ultimate tensile strength and elongation, test specimens are less than 0.08 in. thick.² Hardness, average of ten samples 0.08 in. thick or more.

TABLE 43.—88-8-4 ALLOY

Alloy type	Pounds per square inch		Elongation in 2 in., per cent
	Ultimate tensile strength	Elastic limit	
88-8-4	39,220	11,000	30.6
88-10-2	38,860	12,250	25.2

TABLE 44.—PHOSPHOR BRONZES

No.	Per cent Cu	Per cent Sn	Per cent P	Per cent Fe	Per cent Pb	Per cent Zn	Per cent Ni	Ultimate tensile strength	Elonga- tion
1	95.52	3.87	0.307	0.16					
2	92.5	7.0	0.5						
3	89 max.	11 max.	0.25 max.	0.2 max.	2.25 max.	33,000 ¹	14 ²
4	88.7	11	0.3		35-40,000	6-10
5	85.38	12.55	0.24	0.02	0.61	1.01	0.11	31,000	1.5

¹ MinimaNo. 4 (*Corse*)

Pounds per square inch: Ultimate tensile strength, 35,000-40,000; yield point, 22,000-25,000; compressive elastic limit, 16,000; elastic modulus, 12,000,000-14,000,000.

Per cent: Elongation in 2 in., 6-10; reduction of area, 7-9; Brinell index, 75-85; specific gravity (20°C.), 8.5; Pounds per cubic inch, 0.31; coefficient of friction, 0.004; Shrinkage allowance, $\frac{1}{8}$ in. per foot.

No. 5 (*Lea*)

Pounds per square inch: Ultimate tensile strength, 31,000; elastic limit, 9,800; elastic modulus, 14,800,000; per cent elongation, 1.5.

a minimum elongation in 2 in. of 25 per cent (see Table 40 for required composition). Properties of the cast metal (including endurance limit) as found by Moore in a single series of tests¹ are shown in Table 41.

Sheet-brass requirements are shown in Table 42.

Gun metal (government bronze) is a casting bronze of 86 to 89 per cent copper, 8 to 11 per cent tin, and 1 to 3 per cent zinc; its name of "88-10-2 mixture" defines its type. Its minimum ultimate tensile is 30,000 lb. per sq. in. and minimum elongation (in 2 in.) 14 per cent; it is not recommended for use at temperatures above 260°C.

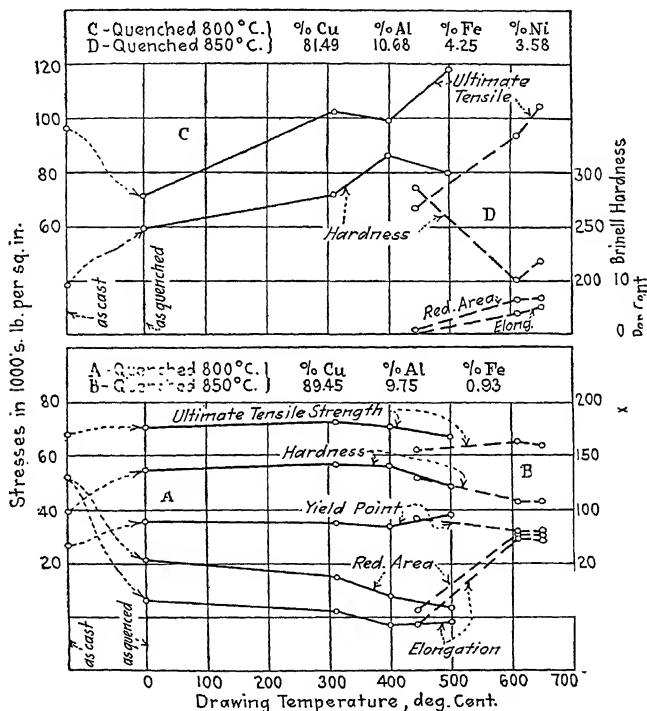


FIG. 25.—Properties of aluminum bronze.

(A.S.T.M., B-10). Burgess and Woodward,² quoting tests by the Bureau of Standards, recommend an 88-8-4 alloy of similar quality; the two alloys are compared in Table 43.

Phosphor-bronze is a generic name for tin bronzes in whose manufacture phosphorus is used as a deoxidizer and in which it remains as a minor ingredient. The copper-rich members of the series are ductile, and a bronze of 98.75 per cent copper, 1.2 per cent tin, and 0.05 per cent phosphorus is suited to wire drawing. Increase in the tin content lowers ductility and augments strength and hardness. In Table 44 alloy No. 1 (95.5 per cent copper) is of rolling grade, and shows torsional ultimate

¹ *Proc. A.S.T.M.*, Vol. 23, part 2, p. 106.

² *Trans. A.I.M.E.*, Vol. 60, p. 179.

strength and elastic limit of 70,000 and 34,200 lb. per sq. in., respectively (Bregowsky and Spring); No. 2 (92.5 per cent copper) is credited with a tensile ultimate of 38,980 lb. per sq. in. and elongation (in 10 in.) of 17.5 per cent; in Nos. 3 4, and 5, of higher tin content, the loss in ductility is evident. Nos. 3 and 4 are gear bronzes, the former specified by the A.S.T.M. (B-22). With the addition of lead, chiefly at the expense of copper, the series extends to the bearing bronzes, for which see Nos. 1 and 2 of Table 52.

Aluminum bronze comprises a series of copper-base alloys containing up to 11 per cent aluminum as an essential constituent; other elements may be added, as in the iron-bearing Sillman bronze. Alloys in the low-aluminum part of this range are very ductile; those of about 10 per cent aluminum are much less so, but have high strength and respond favorably to quenching and drawing, as shown in Fig. 25.¹ Table 45 lists a series of these alloys, of which No. 1 is a 5 per cent aluminum type (Bregowsky and Spring), No. 2 a gear bronze,² and Nos. 3 and 4 a Sillman bronze tested by the Bureau of Standards (No. 3 cast; No. 4, wrought).

TABLE 45.—ALUMINUM BRONZE

No.	Per cent Cu	Per cent Al	Per cent Fe	Per cent Pb	Per cent Sn	Pounds per square inch		Elongation, per cent	Reduction of area, per cent
						Ultimate tensile strength	Elastic limit		
1	94.94	4.90	0.14	0.16	0.03	36,500	28,000	56	47
2	10	1	65,000-80,000	20-30	21-29
3	86.4	9.7	3.9	78,850	11,500	14.5	
3	86.4	9.7	3.9	84,400	14,000	11.5	

No. 2

Pounds per square inch: Ultimate tensile strength, 65,000-80,000; yield point, 23,000-28,000; compressive elastic limit, 19,000; elastic modulus, 15,000,000-18,000,000.

Per cent: Elongation in 2 in., 20-30; reduction of area, 21-29; Brinell index, 92-100; specific gravity (20°C.), 7.5. Pounds per cubic inch, 0.27; coefficient of friction, 0.0025; shrinkage allowance, 0.22 in. per foot.

Zinc.—Unit weight, 0.255 lb. per cu. in.; expansivity, 0.0000311 per degree centigrade at ordinary temperatures. The usual grades and their specified compositions (A.S.T.M., B-6) are shown in Table 46; ornamental castings require a special grade.

TABLE 46.—SPELTER

Per cent maximum	High grade	Intermediate	Brass special	Selected	Prime Western
Pb.....	0.07	0.20	0.60	0.80	1.60
Fe.....	0.03	0.03	0.03	0.04	0.08
Cd.....	0.07	0.50	0.50	0.75	
Pb + Fe + Cd.....	0.10	0.50	1.00	1.25	

Zinc is brittle in tension, plastic in compression, and its strength as cast varies markedly with grain size, hence with relatively slight variation of temperature conditions

¹ CAMPBELL, WILLIAM, Report to Navy Department.

² CORSE, *Trans. A.I.M.M.E.*, Vol. 60, p. 173.

at casting; 4000 to 6000-lb. per sq. in. is a fair average for ultimate tensile strength, 18,000 for a compressive load corresponding to 10 per cent deformation (Riggs and Williams).

Roofing Zinc.—Zinc shingles compete with wood, asbestos, asphalt, tile, and slate, and are cheaper than copper shingles; corrugated zinc is especially recommended as a roofing material in the tropics and semitropics (American Zinc Institute).

Tin.—Unit weight, 0.265 lb. per cu. in.; expansivity, 0.000021 per degree centigrade at ordinary temperatures. The Federal Specification Board's requirements as to composition are shown in Table 47; grade A must be new metal, grade B may be remelted material. Structurally, tin is a roofing material or a means of protective coating, but block tin pipe is also available (Table 48). Ultimate tensile strength, 3500 to 4600; elastic limit, 1500 to 1800; ultimate compressive strength, 6,000; and elastic modulus, 4,000,000 lb. per sq. in. for the cast metal (Carnegie).

TABLE 47.—PIG TIN

Mini- mum per cent Sn	Maxi- mum per cent Pb	Maxi- mum per cent Sb	Maxi- mum per cent Zn	Maxi- mum per cent Cd	Maxi- mum per cent As	Maxi- mum per cent Ca	Maxi- mum per cent S	Maxi- mum per cent Fe	Maxi- mum per cent Bi
99.75	0.10	0.10	None	None	0.10	0.10	0.01	0.01	0.01
98.00	1.50	1.50	None	None	0.10	0.10	0.01	0.10	0.10

TABLE 48.—BLOCK-TIN PIPE (*Hoyt Metal Co.*)

<i>I</i>	<i>O</i>	Ounces per foot	<i>I</i>	<i>O</i>	Ounces per foot	<i>I</i>	<i>O</i>	Ounces per foot
$\frac{1}{8}$	$\frac{3}{16}$	1.0	$\frac{5}{16}$	$\frac{7}{16}$	3.67	$\frac{1}{2}$	$4\frac{3}{64}$	8.0
$\frac{1}{8}$	$\frac{1}{4}$	1.8	$\frac{5}{16}$	$\frac{1}{2}$	5.67	$\frac{1}{2}$	$4\frac{5}{64}$	10.0
$\frac{1}{8}$	$\frac{9}{32}$	2.67	$\frac{5}{16}$	$\frac{9}{16}$	9.0	$\frac{1}{2}$	$\frac{3}{4}$	12.5
$\frac{1}{8}$	$\frac{1}{4}$	3.33	$\frac{3}{8}$	$\frac{7}{16}$	2.0	$\frac{5}{8}$	$2\frac{5}{32}$	9.0
$\frac{3}{16}$	$\frac{1}{4}$	1.0	$\frac{3}{8}$	$\frac{1}{2}$	4.0	$\frac{5}{8}$	$1\frac{3}{16}$	12.0
$\frac{3}{16}$	$\frac{9}{32}$	1.8	$\frac{3}{8}$	$\frac{9}{16}$	7.33	$\frac{5}{8}$	$5\frac{7}{64}$	16.0
$\frac{3}{16}$	$\frac{5}{16}$	2.5	$\frac{3}{8}$	$\frac{5}{8}$	9.67	$\frac{3}{4}$	$5\frac{7}{64}$	9.0
$\frac{3}{16}$	$\frac{3}{8}$	4.33	$\frac{3}{8}$	$1\frac{1}{16}$	12.5	$\frac{3}{4}$	$1\frac{5}{16}$	12.0
$\frac{3}{16}$	$\frac{7}{16}$	6.17	$\frac{7}{16}$	$\frac{1}{2}$	2.0	$\frac{3}{4}$	$6\frac{3}{64}$	16.0
$\frac{1}{4}$	$\frac{9}{32}$	0.75	$\frac{7}{16}$	$\frac{9}{16}$	5.33	1	$1\frac{1}{8}$	12.0
$\frac{1}{4}$	$\frac{1}{4}$	1.33	$\frac{7}{16}$	$\frac{5}{8}$	7.67	1	$1\frac{3}{16}$	16.0
$\frac{1}{4}$	$\frac{3}{8}$	3.17	$\frac{1}{2}$	$1\frac{9}{32}$	4.0	$1\frac{1}{4}$	$1\frac{7}{16}$	20.0
$\frac{1}{4}$	$\frac{7}{16}$	5.17	$\frac{1}{2}$	$\frac{5}{8}$ ^a	5.0	$1\frac{1}{4}$	$1\frac{1}{2}$ ^b	28.0
$\frac{1}{4}$	$\frac{1}{2}$	7.5	$\frac{1}{2}$	$\frac{5}{8}$ ^b	5.5	$1\frac{1}{2}$	$1\frac{11}{16}$	24.0
$\frac{5}{16}$	$\frac{3}{8}$	1.67	$\frac{1}{2}$	$4\frac{1}{64}$	6.0	2	$2\frac{3}{16}$ ^b	32.0

I = inside, *O* = outside diameter in inches.

^a Scant. ^b Full.

Solders.—The lead-tin alloys used as solders are listed in Table 49 (A.S.T.M., B-32); compositions are specified, other data are added as a guide to selection, the melting points of pure lead and pure tin being included as extremes of the series. The alloys are completely solid below "melting point," completely liquid above "complete

liquation point," and partly solid partly liquid between, although in grade 1 the proportion of solid in this range is very slight. In the two alloys of grade 4, the proportion of solid and liquid in the tabular range fits them for use as wiping solders

TABLE 49.—SOLDERS

Grade	Composition, per cent				Temperature, degrees Centigrade	
	Sn	Approximate Pb	Maximum Sb	Maximum Cu	Melting point	Complete liquation point
	100.00	232	232
0A.....	63.00	37.00	0.12	0.08	181	181
1A.....	50.00	50.00	0.12	0.08	181	213
1B.....	49.25	50.00	0.75	0.15	185	208
2A.....	45.00	55.00	0.12	0.08	181	225
2B.....	43.50	55.00	1.50	0.15	188	220
3A.....	40.00	60.00	0.12	0.08	181	237
3B.....	38.00	60.00	2.00	0.15	188	228
4A.....	37.50	62.50	0.12	0.08	181	241
4B.....	35.50	62.50	2.00	0.15	188	231
5A.....	33.00	67.00	0.12	0.08	181	252
5B.....	31.00	67.00	2.00	0.15	188	235
	100.00	327	327

Lead.—Unit weight, 0.41 lb. per cu. in.; expansivity, 0.0000286 per degree centigrade at ordinary temperatures. Table 49 shows the composition limits of pig lead

TABLE 50.—PIG LEAD

Per cent	Corroding	Chemical	Common	
			Desilverized	Soft Missouri
Ag maximum.....	0.0015	0.005–0.015 ^a	0.002	0.002
Cu maximum.....	0.0015	0.04 –0.08 ^a	0.0025	0.04
Cu + Ag maximum.....	0.0025			
As maximum.....	0.0015			
Sb + Sn maximum.....	0.0095			
As + Sb + Sn maximum.			0.015	0.015
Zn maximum.....	0.0015		0.002	0.002
Fe maximum.....	0.002		0.002	0.005
Bi maximum.....	0.05	0.005	0.25	0.005
Pb (by difference).....	99.9330		99.7265	99.931

^a Usual range.

required for usual grades by the A.S.T.M. (B-29); the antimonial grade is treated on page 562. The ordinary metal is marketed as sheet, pipe, tape, wool, etc.; sizes and weights of pipe are listed in Table 51. The ultimate tensile strength of cast lead is about 1800 lb. per sq. in., but increases with mechanical forming to 2200 to 2500 in pipe and wire and 3300 in rolled sheet (Carnegie).

TABLE 51.—LEAD PIPE (*Hoyt Metal Co.*)

I	O	Weight per foot		I	O	Weight per foot		I	O	Weight per foot	
		Pounds	Ounces			Pounds	Ounces			Pounds	Ounces
$\frac{3}{8}$	0.250	0	3		1.076	2	4		2.752	5	0
	0.358	0	4		1.107	2	8		2.893	8	0
	0.381	0	5		1.165	3	0	$2\frac{1}{2}$	3.027	11	0
$\frac{1}{4}$	0.402	0	6		1.221	3	8		3.156	14	0
	0.342	0	8		1.274	4	0		3.280	17	0
	0.511	0	12		1.196	1	10		3.213	5	0
					1.237	2	0		3.244	5	12
	0.522	0	8		1.276	2	6	3	3.374	9	0
	0.553	0	10		1.364	3	4		3.504	12	6
	0.582	0	12		1.435	4	0		3.639	16	0
$\frac{3}{8}$	0.637	1	0		1.503	4	12		3.782	20	0
	0.733	1	8		1.609	6	0				
	0.819	2	0						3.747	6	12
	0.896	2	8		1.446	2	0		3.843	9	8
					1.491	2	8	$3\frac{1}{2}$	4.028	15	0
	0.618	0	8		1.535	3	0		4.142	18	0
	0.670	0	12	$1\frac{1}{4}$	1.599	3	12		4.252	22	0
	0.717	1	0		1.680	4	12				
	0.762	1	4		1.775	6	0		4.1624	5	0
$\frac{1}{2}$	0.804	1	8		1.831	6	12		4.1941	6	0
	0.845	1	12						4.2549	7	15
	0.883	2	0		1.745	3	0	4	4.3949	12	8
	0.955	2	8		1.782	3	8		4.4992	16	0
	1.022	3	0		1.837	4	4		4.6156	20	0
				$1\frac{1}{2}$	1.891	5	0		4.7571	25	0
	0.767	0	12		1.976	6	4		4.8946	30	0
	0.809	1	0		2.090	8	0				
	0.888	1	8		2.153	9	0		5.1438	5	8
	0.959	2	0						5.5470	21	12
$\frac{5}{8}$	0.993	2	4		2.189	3	0	5	5.6182	24	12
	1.026	2	8		2.249	4	0		5.6944	28	0
	1.058	2	12		2.307	5	0		5.7869	32	0
	1.089	3	0	2	2.392	6	8				
	1.148	3	8		2.447	7	8		6.1365	6	4
					2.527	9	0	6	6.4680	22	0
	0.891	0	14		2.623	10	14		6.6100	29	0
$\frac{3}{4}$	0.945	1	4						7.0187	50	0
	1.013	1	12								
				$2\frac{1}{2}$	2.703	4	0				

I = inside, O = outside diameter in inches. For antimonial lead, reduce these weights 4, 5, 6.5, or 8 per cent for antimony contents of 4, 6, 8, or 10 per cent, respectively.

Bearing metals typically are alloys of two or more micro constituents contrasted in hardness: the harder particles provide strength and wearing quality, the softer permit slight adjustment of the wearing surface in service and cause the minute surface inequalities that aid in holding lubricant. Any wide disparity in hardness

between moving part and bearing metal in severe service prevents the mutual polishing of surfaces required in a good bearing, whence the material of the moving part is a factor in selection, as well as in the strength and casting properties of the bearing metal itself.

Bronzes.—In Table 52, grades 1 to 6 have the compositions specified by the A.S.T.M. (B-31) for bronze bearing metal in ingot form; the supplementary data on strength indicate what may be expected of material properly handled. Grades 1 and 2 are phosphor-bronzes; the last two grades, which are liable to segregation of lead, are improved by 1 per cent nickel. For railroad use, metal similar in composition to grades 2 and 3 is specified for eccentric straps, crosshead gibs, etc., and similar to grade 6 for tender, freight, and passenger-car equipment. Alloys A and B are specified (B-22) as to both composition and properties for turntables and swing and lift bridges. They illustrate the choice of bearing metal according to the material of the moving part; A is suited to contact with hardened-steel disks at pressures over 1500 lb. per sq. in. (turntables, swing bridges), B to contact with slow-moving soft-steel parts at lower pressures (trunnions and journals of lift bridges).

TABLE 52.—BEARING BRONZES

No.	Composition, per cent							Pounds per square inch		Elongation in 2 in., per cent	Brinell index
	Cu	Sn	Pb	Maximum Zn	P	Maximum S	Maximum total impurities	Ultimate tensile strength	Compressive deformation limits		
1	85	10	5	0.27	0.70 min.	0.05	0.50	28,000	18,000	12.5	60
2	80	10	10	0.50	0.70 min.	0.05	0.75	25,000	15,000	8	55
3	80	10	10	2.00	0.05 ^a	0.05	2.50	22,000	12,500	8	50
4	77	8	15	0.50	0.25 ^a	0.05	0.75	20,000	12,000	10	48
5	73	7	20	0.50	0.05 ^a	0.25	1.00	18,000	11,000	7	45
6	70	5	25	0.50	None	0.25	1.00	15,000	10,000	5	40
							Unlisted maximum			Permanent set in 1 in. at 100,000 lb. per square inch	
A	b	20 ^a	1.0 ^a	0.50	24,000	0.06–0.12 in.	
B	b	17 ^a	1.0 ^a	0.50	18,000	0.10–0.20 in.	

^a Maximum.

^b Remainder. In grades 1 to 6, Fe is limited to 0.25 per cent, Sb to 0.50 per cent, and Al is excluded.

^c Defined as the load in pounds per square inch that produces 0.001-in. compression in a specimen 1 in. long and 1 sq. in. in section. Permissible variation in specified percentages (except of impurities): not over 5 per cent specified, ± 0.50 ; over 5 to 10 per cent specified, ± 0.75 ; over 10 per cent specified, ± 1.50 .

White Metals.—The original tin-base babbitts have been merged in a series leading to lead-base alloys in which tin plays little or no part. Table 53 shows the 12 compositions specified by the A.S.T.M. (B-23, tentative), to which alloys *a*, *b*, and *c* were added by the War Service Association of Manufacturers of Solder and Bearing Metals; and Table 54, summarizes the classification applied to this series by William A. Cowan. The practice of applying a thin and easily renewed lining of babbitt to the tinned surface of a bronze bearing calls for alloys of great fluidity at pouring temperatures; these are indicated in the table.

Nickel.—Unit weight, 0.319 lb. per cu. in., expansivity, 0.0000132 per degrees centigrade at ordinary temperatures. Alone, or as a prominent member of various

alloys, nickel combines marked resistance to corrosion with strength properties comparable to those of steel. Typical analyses of the metal as produced for fabrication

TABLE 53.—WHITE-METAL BEARING ALLOYS

Grade	Composition, per cent ^a						Temperature, degrees Centigrade		
	Sa	Sb	Pb	Cu	Maximum Fe	Maximum As	Melting point	Complete liquation	Proper pouring
1	91	4.5	0.35 ^b	4.5	0.08	0.10	223	...	411
2	89	7.5	0.35 ^b	3.5	0.08	0.10	241	354	424
3	83.3	8.33	0.35 ^b	8.33	0.08	0.10	240	422	491
4	75	12	10	3	0.08	0.15	184	306	377
5	65	15	18	2	0.08	0.15	181	296	366
a	61.5	10.5	25	3	0.08	0.15			
b	45	7.5	46	1.5	0.08	0.15			
c	30.5	8.5	60	1	0.08	0.15			
6	20	15	33.5	1.5	0.08	0.15	181	277	346
7	10	15	75	0.5 ^b	0.20	240	268	338
8	5	15	80	0.5 ^b	0.20	237	272	341
9	5	10	85	0.5 ^b	0.20	237	256	327
10	2	15	83	0.5 ^c	0.20	242	264	332
11	15	85	0.5 ^b	0.25	244	262	332
12	10	90	0.5 ^b	0.25	245	259	329

Grade	Compression, pounds per square inch						Brinell index	
	Ultimate ^c		Yield point ^d		Elastic limit ^e			
	20°	100°	20°	100°	20°	100°	20°	100°
1	12,850	6,950	4,400	2,650	2,450	1,050	17	8
2	14,900	8,700	6,100	3,000	3,350	1,100	24.5	12
3	17,600	9,900	6,600	3,150	5,350	1,300	27	14.5
4	16,150	6,900	5,550	2,150	3,200	1,550	24.5	12
5	15,050	6,750	5,050	2,150	3,750	1,500	22.5	10
6	14,550	8,050	3,800	2,050	3,550	1,800	21	10.5
7	15,650	6,150	3,550	1,600	2,500	1,350	22.5	10.5
8	15,600	6,150	3,400	1,750	2,650	1,200	20	9.5
9	14,700	5,850	3,400	1,550	2,400	950	19	8.5
10	15,450	5,750	3,350	1,850	2,250	1,200	17.5	9
11	12,800	5,100	3,050	1,400	2,750	1,100	15	7
12	12,900	5,100	2,800	1,250	2,250	950	14.5	6.5

^a Zn and Al, zero. Permitted variations (except in impurities) are ± 0.25 per cent for a specified content of not more than 2 per cent, ± 0.50 per cent over 2 to 5 per cent, ± 0.75 per cent over 5 to 10 per cent, ± 1.00 per cent over 10 per cent. ^b Maximum. ^c Load causing 25 per cent reduction of length in a cylinder specimen 1.5- by 1.5-in. diameter. ^d Load causing 0.125 per cent reduction in gage length. ^e Load corresponding to a rate of deformation 50 per cent greater than in initial load.

are shown in Table 55; cobalt is an invariable minor component (about 0.5 per cent average in the American product) and is usually reported with the nickel content. Table 56 shows the tensile properties of certain fabricated shapes, together with

properties of more general character. Among the commercial forms of nickel are castings up to 2 tons in weight, forged parts, tubes, sheet (some of it electrodeposited), strip, rods, wire, and cold-worked parts. In its familiar use as a plating material, recent advances in the application of heavy coatings (0.01 to 0.02 in., as compared

TABLE 54.—CLASSIFICATION OF WHITE-METAL BEARING ALLOYS (*William A. Cowan*)

Class	Grade ¹	Hard constituents	Soft constituents	Uses
A	1, 2	Sn-Cu compound	Solid solution, Sb in Sn	Almost all uses. Often replaceable by class C.
	3	Sn-Cu compound and SnSb		
B	4-6 and a-c	SnSb and Sn-Cu compound	Eutectoid P ² and related structures	Very thin linings of great fluidity and firm adherence to tinned backs.
C	7	SnSb	Eutectoid O ² and related structures	General service; alloy 7 especially suited to replace class A alloys.
	8, 10	Sb or solution Sn in Sb		
	11	Sb crystals	Pb-Sb eutectic	
D	9	Eutectoid O ²	Soft dendrites of Pb	Unsuited to general service. For special use, as in lining railroad-car bearings.
	12	Pb-Sb eutectic		

¹ See Table 53.

with the customary 0.001 to 0.003) are increasing its protective value. It can be welded by oxyacetylene or electric methods; under special rolling conditions it can also be plastically welded to steel.

TABLE 55.—MALLEABLE NICKEL (*International Nickel Co.*)

Per cent	Electro	A	B	C
Ni and Co.	99.40	99.00	98.75	96.75
Co.	0.80			
Fe.	0.30	0.55	0.50	0.75
C.	0.10	0.15	0.15	0.15
S.	0.015	0.025	0.025	0.03
Si.	0.10	0.10	0.20	0.20
Mn.	0.15	1.75	1.75

Chrome-nickel alloys (26-30 Cr, 8-20 Ni; or 35-40 Ni, 15-18 Cr; or 60-70 Ni, 12-20 Cr, balance iron) are resistant to oxidation up to about 2000°F. These alloys are subject to intergranular corrosion, but can be stabilized to a large extent by Va and Cb.

Nickel-copper alloys include both structural and special heat- and corrosion-resisting types, of which the latter are treated in the third section of this chapter.

Monel Metal.—Unit weight, 0.318 lb. per cu. in.; expansivity, 0.000014 per degree centigrade at ordinary temperatures. This natural alloy contains roughly 67 per cent nickel, 28 per cent copper, and 5 per cent of other elements, and in strength and resistance to corrosion resembles nickel itself. Table 57 quotes Navy Department specifications for the cast and hot-rolled or forged metal, together with other strength data drawn from various sources. Monel metal is a standard structural material, not only under ordinary rusting and steam service conditions, but also for specialized chemical and metallurgical construction. In the former field it is used for pump liners and rods, valve fittings, mining machinery, marine construction, roofing, and in a variety of other ways. It is cast, rolled, drawn, spun, and stamped in all the usual fabricated forms and can be welded by oxyacetylene or electric methods.

TABLE 56.—PROPERTIES OF NICKEL

Tensile properties (Grade A, Table 55)	Pounds per square inch		Elongation 2 in., per cent	Reduction of area, per cent
	Ultimate tensile strength	Yield point		
Hot-rolled rods.....	70,000– 80,000	20,000– 30,000	40–50	50–70
Sheet:				
Hard-rolled.....	90,000–110,000	85,000–105,000	1–2	20–40
Annealed.....	60,000– 75,000	15,000– 25,000	35–45	40–60
Wire:				
Hard-drawn.....	90,000–110,000	85,000–105,000		
Annealed.....	65,000– 75,000	20,000– 30,000		
Cast nickel (deoxidized)..	50,000– 60,000	20,000– 30,000	20–30	

Hardness, Brinell index:

 Cold-worked..... 130–160

 Annealed..... 80–100

Elastic modulus:

 In tension..... 30,000,000–33,000,000 lb. per square inch

 In shear..... 10,000,000 lb. per square inch

Modified Monel metal ("M-M-M"), developed for use in valves and turbines in superheated-steam service, contains 60 to 65 per cent nickel, 24 to 27 per cent copper, and 9 to 11 per cent tin. Its strength properties are shown in Table 58 (Manning, Maxwell, and Moore). It is easily cast, even in sections as thin as $\frac{1}{8}$ in.

Cupronickel, Benedict, and Related Alloys.—Table 59 shows compositions and tensile properties of important members of the copper-nickel series. The 2.5 per cent nickel alloy is used in the driving bands of shells, that containing 14 to 16 per cent nickel (Benedict metal) in condenser tubes and feed-water heaters, the 25 per cent nickel alloy in subsidiary coinage, the 40 to 45 per cent nickel alloy (Constantan) in pyrometer and electrical-resistance wire. These alloys are free from "season" cracking.

Ambac is an alloy of 75 per cent copper, 20 per cent nickel, and 5 per cent zinc, the last component acting as a deoxidizer to improve working qualities. Table 60 shows tensile properties; the metal is easily rolled, forged, drawn, and cold-worked,

and has good corrosion-resisting properties; it can be cold-drawn into fine wire without becoming embrittled, and is free from "season" cracking (American Brass Co.).

TABLE 57.—MONEL METAL

<i>Composition</i>	Minimum per cent Cu	Minimum per cent Ni	Maximum per cent Fe	Maximum per cent Al	Maximum per cent Mn	Maximum per cent C + Si
	23	60	3.5	0.5	3.5	2.0
Hot-rolled or forged ^a	23	60	3.5	0.5	3.5	0.8
	Pounds per square inch			Elonga- tion, per cent	Brinell index	Cold bend <i>d = t</i>
	Ultimate	Yield point	Proportional limit			
<i>Tension</i>						
Cast.....	65,000 ^a	32,500 ^a	15,000-25,000	25 ^a	100-120	
Rods and bars, round, square:						
1 in. or less.....	80,000 ^a	40,000 ^a	32,000-37,000	30 ^a	180 ^a
1½-2 in.....	85,000 ^a	48,000 ^a		30 ^a	180 ^a
2½-3½ in.....	80,000 ^a	38,000 ^a	} hot rolled	35 ^a	180 ^a
Over 3½ in.....	80,000 ^a	40,000 ^a		32 ^a	180 ^a
Hexagons and rectangles.....	80,000 ^a	40,000 ^a		32 ^a	180 ^a
Sheets and plates.....	65,000	30,000 ^a		15 ^a	120 ^a
Cold-drawn rods.....	80,000-110,000	60,000- 90,000		20-30	150-170	
Sheet:						
Hard.....	100,000-120,000	90,000-110,000		1- 2	180-200	
Annealed.....	65,000- 75,000	25,000- 35,000		35-45		
Wire:						
Hard.....	100,000-120,000	90,000-110,000		1- 2		
Annealed.....	65,000- 75,000	25,000- 35,000		20-30		
				Shear, inch per inch ^b	Elastic modulus, pound per square inch	
<i>Torsion</i>						
Hot-rolled, rounds:						
¾-1 in.....	62,000	30,000	25,200	2.78	9,240,000	
1¼-2¼ in.....	61,800	38,600	29,700	2.83	9,640,000	
2½-3½ in.....	63,300	26,700	20,200	2.64	9,620,000	
Hot-hammered:						
Rounds, ¾-6 in.....	62,100	32,200	24,200	2.57	9,750,000	
Hexagon ¾-2 in.....	63,600	36,000	24,800	1.55	9,470,000	
<i>Shear</i>						
Hot-rolled rods:						
¾-in., single shear.....	45,000- 50,000			Tensile elastic modulus 25,000,000-26,000,000 lb. per square inch	
double shear.....	90,000-103,000				
1-in., single shear.....	55,000- 61,000				
double shear.....	115,000-127,000				
<i>Compression</i>						
Cast.....		20,000- 30,000	15,000-25,000			
Rods.....		60,000- 70,000	25,000-50,000			

^a U.S. Navy specifications, minimum as to strength properties.

^b Of gage length.

Admic ("Admiralty nickel," *i.e.*, the customary copper-zinc-tin Admiralty metal with zinc replaced by nickel) contains 70 per cent copper, 29 per cent nickel, and 1 per

cent tin. It was developed as a material for diaphragms, especially in thermostats, and in this service has withstood between one and two million cycles of operation with negligible permanent set and softening. A $\frac{1}{2}$ -in. hard rod showed elastic modulus of 21,200,000; ultimate tensile strength, yield point, and elastic limit of 112,500, 108,500,

TABLE 58.—MODIFIED MONEL METAL

Temperature degrees Centigrade	Tension				Compression, elastic limit, pounds per square inch
	Ultimate, pounds per square inch	Elastic limit, pounds square inch	Elastic in 2 in., per cent	Reduction of area, per cent	
20	70,200	43,500	5	4	40,000
316	67,500	46,000	2	3.5	
482	59,150	43,900	2	3.5	

TABLE 59.—COPPER-NICKEL ALLOYS

Composition per cent		Form	Pounds per square inch		Elongation, per cent
Cu	Ni		Ultimate tensile strength	Yield point	
98	2	Tubes:			
		Soft.....	39,200	44
		Hard.....	63,500	5.5
97.5	2.5	Driving bands.....	31,770	14,340	69.6
		Wire (0.08-in. diameter), hard.....	72,150	1.6
90	10	Strip, annealed.....	48,450	28.4
84-86	16-14	Benedict metal tubes:			
		Diameter			
		Outside Inside			
		Annealed..... 0.517 0.441	43,080	39.0
		Light drawn..... 0.516 0.444	51,520	8.1
		Medium drawn..... 0.750 0.656	60,300	4.2
75	25	Hard drawn..... 0.750 0.656	75,800	2.0
		$\frac{5}{8}$ -in. rod:			
		Hard.....	64,000	58,000	17
55	45	Annealed.....	51,600	21,000	39
		$\frac{1}{4}$ -in. hot-rolled rod.....	65,000- 85,000	25-40
		Annealed wire.....	60,000- 65,000		
		Hard-drawn wire.....	110,000-140,000		

and 85,000, respectively (all in pounds per square inch); elongation of 12.5 per cent and reduction of area of 53 per cent (data from W. B. Price, Scoville Mfg. Co.).

Aluminum.—Unit weight, 0.0975 lb. per cu. in.; expansivity, 0.00002353 per degree centigrade at ordinary temperature. The physical properties of the metal are discussed in detail in Chap. I of the volume entitled "Recovery of the Metals";

TABLE 60.—AMBRAC

Tension	Pounds per square inch		Elongation in 2 in., per cent
	Ultimate tensile strength	Yield point	
Sheet:			
Soft.....	50,000	24,000	30
Light rolled.....	60,000	50,000	20
Hard rolled.....	84,000	2.5
Hard rod:			
$\frac{1}{2}$ – $\frac{3}{8}$ in.....	80,000	12
$\frac{7}{16}$ in.....	75,000	65,000	12
$1\frac{1}{4}$ – $1\frac{1}{2}$ in.....	65,000	60,000	15

TABLE 61.—TENSILE PROPERTIES OF SUBSTANTIALLY PURE ALUMINUM¹

Form	Tensile strength, pounds per square inch	Yield point, pounds per square inch	Elongation on a 2-in. length, per cent	Reduction in area, per cent
Sand cast.....	11,000–13,000	8,000– 9,000	15–25	30–40
Chill cast.....	12,000–14,000	9,000–10,000	20–30	50–60
Sheet:				
Fully annealed.....	12,000–15,000	8,000– 9,000	15–30	30–40
Half hard.....	18,000–22,000	10,000–12,000	10–20	20–30
Hard (heavy reduction)...	22,000–35,000	12,000–25,000	2–10	5–20
12-gage (ordinary reduction).....	25,000	17,000		15
16-gage (ordinary reduction).....	28,000	20,000		10
20-gage (ordinary reduction).....	30,000	22,000	3	7
Bar, hard-drawn.....	28,000–35,000	14,000–25,000	3–10	6–20
Wire:				
Hard-drawn.....	22,000–55,000	14,000–33,000	1–10	2–20
40-mil.....	31,000	23,000	3	6
80-mil.....	28,000	20,000	5	10
120-mil.....	25,000	16,000	7	15
200-mil.....	22,000	14,000	0	20

¹ Approximate values, from various sources.

of these, the most distinctive is its lightness, which is largely shared by the aluminum-base alloys, some of them with strength comparable to that of mild steel. Aluminum can be worked hot or cold, stamped, spun, rolled, drawn, and otherwise fabricated in the usual commercial forms. Table 61 (from Chap. I of the volume entitled "Recovery of the Metals") shows its tensile properties in the more common of these, and Table 62 contains data on minimum strength and approximate hardness of aluminum sheet, rolled from metal of 99 per cent grade or better. The elastic modulus of the cast metal is 10,000,000 and its ultimate compressive strength 60,000 lb. per sq. in.; its Brinell index is about 25 (Table 6, Chap. I of the volume entitled "Recovery of the Metals").

TABLE 62.—FLAT AND COILED ALUMINUM SHEET (*Aluminum Co. of America*)

Temper	Ultimate tensile strength, pounds per square inch	Per cent elongation in 2 in., for B. & S. gage Nos.							Brinell index
		2-4	5-6	7-9	10-16	17-20	21-24	25-28	
Soft.....	12,000	30	30	30	30	25	20	15	22
Quarter hard.....	14,000	..	10	10	9	7	5	..	27
Half hard.....	16,000	7	7	5	4	3	32
Three-quarters hard.....	18,000	5	3	2	2	35
	19,000	4	3	2	1	37
	20,000	4	3	2	1	39
	22,000	4	40
Hard.....	24,000	3	43
	26,000	2	..	46
	30,000	1	50

Aluminum-base Alloys.—For general discussion, see Chap. I of the volume entitled "Recovery of the Metals."

Casting alloys have been developed in great number, of which the physical properties of a representative series are shown in Table 8 of that chapter. Table 63 presents data on sand-cast alloys commonly used in American practice. Numbers 43, 45, and

TABLE 63.—SAND-CAST ALUMINUM ALLOYS (*Aluminum Co. of America*)

No.	Composition, per cent						Ultimate tensile strength ¹	Elongation in 2 in., per cent	Approximate	
	Al	Cu	Fe	Zn	Si	Mn			Yield point ¹	Brinell
100	99 min.	12,000-14,000	15-25	4,000	25
12	92	8	18,000-23,000	1-3	10,000	65
112	Bal.	6-8	1.5 max.	2.5 max.	19,000-24,000	1-2.5	11,000	65
109	Bal.	12.5	19,000-26,000	0-1.5	15,000	70
43	Bal.	...	1.0 max.	...	5	...	17,000-21,000	3-7	7,000	40
45	Bal.	10	...	19,000-22,000	1-3	9,000	50
47 ²	Bal.	12.5	...	25,000-30,000	5-10	11,000	60
106	Bal.	2	...	16,000-20,000	3.5-6.5	6,000	40
195	Bal.	4	28,000-35,000	5.5-11	18,000	70
196	Bal.	5	36,000-45,000	0-2.5	115
145	Bal.	2.5	1.25	10	25,500-34,000	3-6	12,000	65

¹ Pounds per square inch. ² Modified alloy.

47 are especially recommended for thin sections; they are leakproof, have low shrinkage, and (with No. 106) show good resistance to relatively severe forms of atmospheric corrosion. Castings of alloys 195 and 196 may be heat-treated.

Wrought Alloys.—For certain casting alloys used in wrought form, see Chap. I of the volume entitled "Recovery of the Metals." Table 64 contains data on sheet rolled from an aluminum alloy containing about 1.5 per cent manganese. The standard members of this group comprise duralumin and two newer combinations described by Archer and Jeffries.¹ All three are hardened by quenching from about 500°C. and acquire further strength and hardness by subsequent "aging," to which, however, each has its own separate response. Duralumin (the copper-magnesium alloy, represented by 17S in the table), hardens spontaneously at room temperature

TABLE 64.—FLAT AND COILED ALUMINUM-ALLOY SHEET (*Aluminum Co. of America*)

Temper	Ultimate tensile, strength, pounds per square inch	Per cent elongation in 2 in., for B. & S. gage Nos.						Brinell index
		2-4	5-6	7-9	10-16	17-20	21-24	25-28
Soft.....	16,000	25	25	25	20	20	15	10
Quarter hard.....	19,000	..	8	8	7	6	5	..
Half hard.....	21,000	6	6	4	3	2
Three-quarters hard.....	23,000	4	3	2	2
	25,000	4	3	2	1
	26,000	4	3	2	1
	27,000	4
	30,000	3
Hard.....	33,000	2	..
	35,000	1

Elongation are minima with or across grain for the "soft" and "hard" tempers, with grain only for intermediate tempers. Maximum thickness of coiled sheet, No. 10 B. & S.

for several days after the quench; the magnesium-silicon alloy (No. 51-S) similarly hardens to some extent at room temperature, more markedly if heated at 100 to 175°C.; the copper alloy (No. 25S) can be age hardened only by heating to 100 to 175°C. Aside from quenching, all three alloys respond to various forms of treatment. Annealing reduces strength and hardness and permits the maximum of mechanical forming, which, in turn, hardens and strengthens the material at the sacrifice of ductility. Quenching and aging, on the other hand, promote strength and hardness without impairing ductility. Extrusion, as a form of hot working, has an incidental effect of heat-treatment, of which the results on physical properties vary with manufacturing conditions, and hence can be illustrated by general values only. Tables 65 and 66 show the properties of these alloys in various conditions, subject, in the case of heat-treated material, to prior obliteration of the cast structure by mechanical work. Due to its spontaneous aging, the data for No. 17-S "quenched and unaged" are applicable for only 1 to 4 hr. after the quench; during this time it withstands mechanical forming almost as well as when annealed, but later rapidly hardens and becomes more difficult to work. Of the three alloys, duralumin is not so easily worked as either 25-S or 51-S; of these two, 25-S is the stronger, 51-S the better fitted for severe cold forming, although, if yield point rather than tensile strength is the requisite, the two alloys when properly treated differ very little. Number 25-S has been extruded in the smaller I-beam and channel sections for structural use.

¹ *Trans. A.I.M.E.*, 1925.

TABLE 65.—DURALUMIN (*Alloy 17-S, Aluminum Co. of America*)
General Properties

		Composition, per cent					Tension, pounds per square inch		Elongation in 2 in., per cent	Brinell index
		Cu	Mg	Mn	Al		Ultimate tensile strength	Yield point		
17S {	Minimum .	3.5	0.2	0.4	92	Hard rolled.....	43,000	30,000	4	80
	Maximum .	4.5	0.75	1.0	...	Annealed.....	30,000	9,000	18	54
	Nominal...	4	0.5	0.5	Bal.	Quenched and unaged.....	48,000	20	78
A17S nominal...		2.5	0.3	0	Bal.	Quenched and aged.....	60,000	33,000	20	95
B17S nominal ..		3.5	0.3	0	Bal.					

Selected Forms

Temper	Sizes, inches	Tension, pounds per square inch		Elongation in 2 in., per cent	Cold bend d =
		Ultimate	Yield point		
Sheet:					
17S Annealed.....	0.500-0.250	25,000-35,000	12-22	3-2t
	<0.250-0.125	25,000-35,000	12-22	1.5-0.5t
	<0.125-0.021	25,000-35,000	10-12	0.5-0t
	0.020-0.013	25,000-35,000	8-18	0
17S Quenched and aged..	0.500-0.250	50,000-55,000	25,000-33,000	12-20	4-3t
	<0.250-0.125	53,000-60,000	28,000-35,000	15-22	3-2.25t
	<0.125-0.020	55,000-65,000	30,000-38,000	18-25	2.25-2t
	0.019-0.013	55,000-65,000	30,000-38,000	15-22	2t
A17S Annealed.....	Intermediate	20,000-25,000	20-28	
A17S Quenched and aged ¹ ..	Intermediate	35,000-45,000	15,000-20,000	20-28	
B17S Annealed.....	Intermediate	22,000-30,000	20-28	
B17S Quenched and aged ² ..	Intermediate	42,000-50,000	20,000-25,000	20-28	
Bar:					
17S Annealed.....	0.75-1.5	35,000 max.	16 min. ³	
17S Quenched and aged..	Over 1.5	45,000-55,000	20,000-30,000	18-25	
	>0.75-1.5	50,000-60,000	25,000-35,000	16-23	
	0.125-0.75	55,000-65,000	25,000-35,000	14-22	2t
Tubing:					
17S Quenched and aged..	0.25-1.0	55,000-65,000	30,000-40,000	18-28	
	>1.0-1.5	55,000-65,000	30,000-40,000	16-23	
	>1.5-3.0	55,000-65,000	30,000-40,000	14-20	

175S, Quenched and Aged

Shear strength, 30,000-35,000 pounds per square inch; compressive strength equal to tensile; specific gravity, 2.76-2.82 (not more than 2.85); weight, pounds per cubic inch, 0.101 to 1.103; melting range, 540 to 650°C.; expansivity, 0.000023 per degree Centigrade (20 to 100°C.); elastic modulus in tension, 10,000,000 pounds per square inch; endurance limit, 10,000 to 12,000 pounds per square inch.

¹ Brinell index 55 to 75. ² 65-85. ³ Elongation in 4t.

Cadmium.—Apart from the great value of cadmium in fusible alloys, which hardly come under Materials of Metallurgical Construction, the war has demonstrated the value of cadmium plating where it is necessary to protect against exposure to corrosion. It appears to be vastly superior to zinc, in spite of the close resemblance between the two metals chemically.

TABLE 66.—AL-CU AND AL-MG-SI ALLOYS (Nos. 25S and 51S, *Aluminum Co. of America*)

Tension	Aluminum-copper type				Aluminum-magnesium-silicon type			
	Ultimate tensile strength, pounds per square inch	Yield point, pounds per square inch	Elongation in 2 in., per cent	Brinell index	Ultimate tensile strength, pounds per square inch	Yield point, pounds per square inch	Elongation in 2 in., per cent	Brinell index
Annealed.....	25,000-35,000	15-20	45-55	14,000-18,000	15-30	23-30
Extruded.....	30,000-40,000	15,000-20,000	10-18	50-65	20,000-30,000	9,000-13,000	18-25	40-55
Quenched ¹	45,000-53,000	18,000-25,000	15-20	70-85	30,000-40,000	15,000-20,000	20-30	55-70
Fully treated ²	55,000-63,000	30,000-40,000	18-25	90-110	45,000-50,000	30,000-40,000	10-18	85-100

¹ From 515 to 540°C.

² For average use; heated (after quenching) for 4 to 18 hr. at 100 to 160°C.

Fusible Alloys.—The following table of alloys fusible at comparatively low temperatures is taken from Johnson's "Materials of Construction," fifth edition. See also the chapter on Bismuth in the volume entitled "Recovery of the Metals."

TABLE 67.—ALLOYS FUSIBLE AT LOW TEMPERATURE

Name	Per cent Bi	Per cent Pb	Per cent Sn	Per cent Cd	Fusion temperature, degrees Centigrade
Newton's.....	50	31	19	0	95
Rose's.....	50	28	22	0	100
Darcel's.....	50	25	25	0	93
Wood's.....	50	24	14	12	66-71
Lipowitz's.....	50	27	13	10	60

CEMENT, CONCRETE, STONE, AND CLAY PRODUCTS

Portland cement is the finely pulverized clinker "produced by calcining to incipient fusion an intimate and properly proportioned mixture of argillaceous and calcareous materials, with no additions subsequent to calcination excepting water and calcined gypsum." This definition and the succeeding specifications are those of the A.S.T.M. (C-9).

Chemical Requirements.

	Maximum, per cent	Permissible variation, per cent
Loss on ignition.....	4.00	0.25
Insoluble residue.....	0.85	0.15
Sulphuric anhydride.....	2.00	0.10
Magnesia.....	5.00	0.40

A result exceeding the specified maximum by not more than its permissible variation is reported at the maximum figure (see Chemical Analysis, below).

Physical Requirements.—Minimum specific gravity (if specified), 3.10; for white Portland cement, 3.07. Maximum oversize on a standard No. 200 sieve, 22 per cent by weight. Neat cement paste of normal consistency (see Test Mixtures) is spread on glass and drawn together in a pat 3 in. in diameter, $\frac{1}{2}$ in. thick at center and tapered to a thin edge; after storage in moist air for 24 hr. and exposure to an atmosphere of steam at 98 to 100°C. for 5 hr., the pat must show no signs of distortion, cracking, checking, or disintegration (shrinkage cracks formed in the first 24 hr. excepted). Minimum period (after mixing) to reach initial set as defined by the Vicat needle, 45 min.; by the Gillmore needle, 60 min. (see Time of Set). Maximum period to reach final set, 10 hr. Minimum ultimate tensile strength (pounds per square inch), average of three standard mortar briquettes: after one day's storage in moist air, 6 days in water, 200; after one day in moist air, 27 days in water, 300; strength of standard mortar at 28 days must exceed that at 7 days.

Chemical Analysis of Cement.—The following quick ("purity") test is drawn from Johnson's "Materials of Construction," fifth edition. Place half a teaspoonful of cement in a test tube and cover with a 50 per cent solution of hydrochloric acid, stirring with a glass rod. Good cement effervesces slightly and forms a yellow or orange-yellow jelly. Of the adulterants, calcium carbonate causes violent effervescence, slag evolves hydrogen sulphide, and others produce a dark sediment at the bottom of the jelly. The A.S.T.M. specifications are resumed below.

Loss of Ignition.—Put 1 g. of cement in a covered platinum crucible (20 to 25 cc.) and either (a) blast for 15 min. at full red heat with inclined flame, holding the crucible fixed in an asbestos board; or (b) heat in a muffle for 15 min. at 900 to 1000°C. In each case check the loss in weight by igniting for an additional 5 min.

Insoluble Residue.—Treat 1 g. of cement with 10 cc. of water and 5 cc. of concentrated HCl; warm until effervescence ceases. Dilute to 50 cc., and digest on a steam bath or hot plate until decomposition of the cement is complete. Filter (1), wash with cold water, and digest filter paper and contents in about 30 cc. of a 5 per cent solution of sodium carbonate maintained at a temperature just below boiling for 15 min. Filter the remaining residue, wash with cold water, then with a few drops of hot HCl, 1:9, then with hot water; ignite at red heat and weigh.

Sulphuric Anhydride.—Either (a) dissolve 1 g. of cement in 5 cc. of concentrated HCl and 5 cc. of water, warming gently, and when solution is complete add 40 cc. of water, filter, and wash thoroughly; or (b) use filtrate (1) of the insoluble-residue test. Dilute the filtrate to 250 cc., heat to boiling, and add 10 cc. of hot 10 per cent barium chloride solution drop by drop; continue boiling until the precipitate is well formed, then digest on the steam bath until it settles. Filter and wash, place filter paper

and contents in a platinum crucible, and heat so that the paper is consumed without flaming, then ignite and weigh. [Weight of barium sulphate] $\times 34.3$ = per cent sulphuric anhydride. (The method gives results a little low, as some BaSO_4 is converted to BaS by the ignition of the filter paper.)

Magnesia.—(a) Place 0.5 g. of cement in an evaporating dish, add 10 cc. of water, then 10 cc. of concentrated HCl; heat gently and agitate until reaction ceases. Evaporate to dryness on a steam or water bath; to hasten dehydration the residue may be heated for 30 to 60 min. at 150 to 200°C. Treat the residue with 10 cc. of concentrated HCl and 10 cc. of water; digest for 10 min. in a covered dish on steam or water bath, dilute, filter, and thoroughly wash the separated silica. (b) To the filtrate (about 250 cc.) add 5 cc. of concentrated HCl and enough bromine water to precipitate any manganese; add ammonium hydroxide until alkaline, boil until only a faint odor of ammonia remains, wash the precipitated iron and aluminum hydroxides once by decantation, and filter (1), washing slightly on the filter. Return the precipitate to the precipitating vessel, and dissolve in 10 cc. of hot HCl, extracting the paper with acid and combining this solution with that of the bulk precipitate. Reprecipitate at boiling heat by ammonium hydroxide and bromine water (total volume of solution about 100 cc.), filter (2) and wash. (c) Combine filtrates (1) and (2), concentrate if necessary, add 1 cc. of ammonium hydroxide, bring to boiling, add 25 cc. of a saturated solution of boiling ammonium oxalate, and continue to boil until the calcium oxalate precipitate is granular. After an hour filter (3) and wash, place filter paper and contents wet in a platinum crucible and burn off the paper; after ignition redissolve in HCl, dilute to 100 cc., add a slight excess of ammonia and boil. Reprecipitate by ammonium oxalate, settle, filter (4), and wash. (d) Combine filtrates (3) and (4), acidify with HCl, concentrate on a steam bath to about 150 cc. and make slightly alkaline with ammonium hydroxide, boil and filter (to remove a little iron and aluminum—perhaps also calcium). When cool, add 10 cc. of a saturated solution of sodium ammonium hydrogen phosphate to the filtrate; when the ammonium magnesium orthophosphate has formed, add ammonia in excess, and keep for several hours in a cool place.¹ Filter and wash with water containing 2.5 per cent ammonia. Dissolve the precipitate in a little hot HCl, dilute to 100 cc., add 1 cc. of the sodium ammonium hydrogen phosphate as before and ammonia in excess. Let stand for 2 hr., filter and wash as before. Carefully burn off the filter paper in a platinum crucible and ignite to constant weight at a temperature not high enough to soften the magnesium pyrophosphate. The weight of the latter, multiplied by 72.5, equals per cent of magnesia.

Physical Testing of Cement.—Various tests have been developed, among them compression and bending tests, and determinations of porosity, permeability, and adhesion. The following treatment follows the A.S.T.M. specifications.

Specific Gravity.—Fill a standardized Le Châtelier apparatus with kerosene free from water, or benzine of not less than 62°Bé., to a reading between 0 and 1 cc. in the stem. Immerse in water (to avoid variations greater than 0.5°C. in the temperature of the liquid) and slowly introduce 64 g. of cement at the same temperature, rolling the flask to expel entrained air.

$$\text{Specific gravity} = \frac{\text{weight of cement in grams}}{\text{displaced volume (= volume increase) in cubic centimeters}}$$

Duplicate results should agree within 0.01.

Fineness.—Dimensions of the standard No. 200 sieve are shown in Table 68; maximum permitted variation of the sieve, as tested with standardized material of

¹ To obtain best results, stir vigorously, but so far as possible avoid allowing the stirring rod to touch the beaker.—Editor.

20 to 25 per cent oversize, 1.5 per cent. In testing cement for fineness, use a 50-g. sample, slightly incline the sieve, and strike it against the palm of the hand (on the upstroke) about 150 times a minute, rotating it 60 deg. every 25 strokes. The test ends when 1 min. of operation yields not more than 0.05 g. of undersize.

TABLE 68.—DIMENSIONS OF STANDARD SIEVES
Dimensions in inches

No.	Number of wires per whole inch		Limiting diameter of wire
	Warp wires	Shoot wires	
20	19.5-20.5	19-21	0.0160-0.0170
30	29.5-30.5	28.5-31.5	0.0105-0.0115
100	95-101	93-103	0.0042-0.0048
200*	192-208		0.0019-0.0023

* Maximum aperture, 0.0050.

Test Mixtures and Normal Consistency.—In mixing, place the weighed dry materials on a nonabsorbent surface and (for mortar) first mix dry; then form a crater, pour in the measured quantity of water, and trowel inward from the edges of the heap. After 30 sec. (for absorption of water) mix and knead with the hands for at least a minute; too much kneading is better than too little. The room and mixing water should be at 70°F. (20°C.); 500 to 1000 g. of dry material should be mixed in a batch.

TABLE 69.—TABLE SHOWING PERCENTAGE OF WATER REQUIRED FOR STANDARD MORTAR CORRESPONDING TO VARYING PERCENTAGE OF WATER NECESSARY FOR NEAT CEMENT MIXTURES

Neat Cement Paste, Normal Consistency, Per Cent Water	Standard Mortar, 1 Part Cement, 3 Parts Ottawa Sand, Per Cent Water
15	9.0
16	9.2
17	9.3
18	9.5
19	9.7
20	9.8
21	10.0
22	10.2
23	10.3
24	10.5
25	10.7
26	10.8
27	11.0
28	11.2
29	11.3
30	11.5

To find the right proportion of water for "normal consistency," trial pastes with varying proportions are tested with the Vicat apparatus. This consists of a graduated rod (300 g.), sliding vertically in a standard equipped with set screw; one end of the rod is 1 cm., the other 1 mm. in diameter. A 500-g. sample is mixed as described, formed into a ball, tossed six times between the hands (held about 6 in. apart), and then pressed into a taper rubber ring 7 cm. in diameter at the base and 4 cm. high. The base is freed of excess cement by a single sweep of the palm, then placed on a glass plate, and the excess cement at the smaller end (testing surface) is sliced off with one stroke of the trowel; compression of the paste is avoided throughout. The 1-cm. end of the rod is brought to the surface of the paste, read, and quickly released; normal consistency requires that it shall sink 10 mm. below the surface in $\frac{1}{2}$ min.¹ To convert the right proportion of water for neat cement into that for standard mortar, use Table 69. Standard mortar is a mixture of 1 part cement and 3 parts natural sand from Ottawa, Ill. (Ottawa Silica Co.); the sand, sized through a No. 20 sieve, is standard if not more than 5 g. of a 500-g. sample passes a No. 30 in 1 min. of sizing (see Table 68).

Time of set is determined as required, by either the Vicat or the Gillmore apparatus. The latter consists of two weighted needles sliding vertically in a standard; a cement-

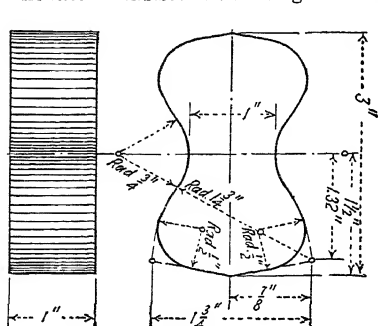


FIG. 26.—Cement test briquette.

sample. During test, the samples are kept in moist air (over water or in a moist closet).

Tensile Test.—Figure 26 shows the standard form of briquette. Wipe the mold with an oily cloth, fill with standard mortar (see Test Mixtures), press firmly, and smooth with the trowel, but avoid ramming; heap more mortar over the mold and smooth with the trowel, using moderate pressure. Turn the mold over, heap, press, and smooth again. When aged as specified, break the briquette in any standard machine, using a rate of loading of 600 lb. per min. Discard any result obtained from a faulty briquette, or that differs more than 15 per cent from the average of the batch.

Miscellaneous.—In the soundness test with a neat cement pat, special means are required to provide storage at 98 to 100°C. in an atmosphere of steam—this is easily designed to utilize laboratory sources of heat. The moist closet is a soapstone, slate, concrete, or metal-lined wooden box, of which the bottom is kept covered with water. Test pieces, unless otherwise specified, are stored in the closet for 20 to 24 hr. after molding.

¹ The use of the Dow consistency needles is far easier and better.—EDDROX.

Natural cement (A.S.T.M. C-10) is the finely pulverized product of calcination of an argillaceous limestone at a temperature only sufficient to drive off carbonic acid gas. No chemical requirements are imposed.

Physical Tests.—Maximum oversize on a standard No. 100 sieve, 10 per cent; on a standard No. 200, 30 per cent by weight (see Table 68). Pats of the dimensions specified for Portland cement are kept under frequent observation for at least 28 days, one in air at normal temperature, another in water at 70°F.; they must remain firm and show no signs of distortion, cracking, checking, or disintegration. As determined by the Vicat needle (see Portland Cement), the cement shall reach initial set in not less than 10 min., and final set in not less than 30 min. nor more than 3 hr. after mixing. Minimum ultimate tensile strength (pounds per square inch) determined as for Portland samples:

	Neat cement	1 part cement, 3 parts standard Ottawa sand
24 hr. in moist air	75	
7 days (one in moist air, six in water)	150	50
28 days (one in moist air, 27 in water)	250	125

Methods of Test.—As for Portland cement.

Miscellaneous cements are of minor industrial importance, but their use is occasionally indicated by either convenience or economy. The subjoined discussion of concrete presupposes standard Portland cement.

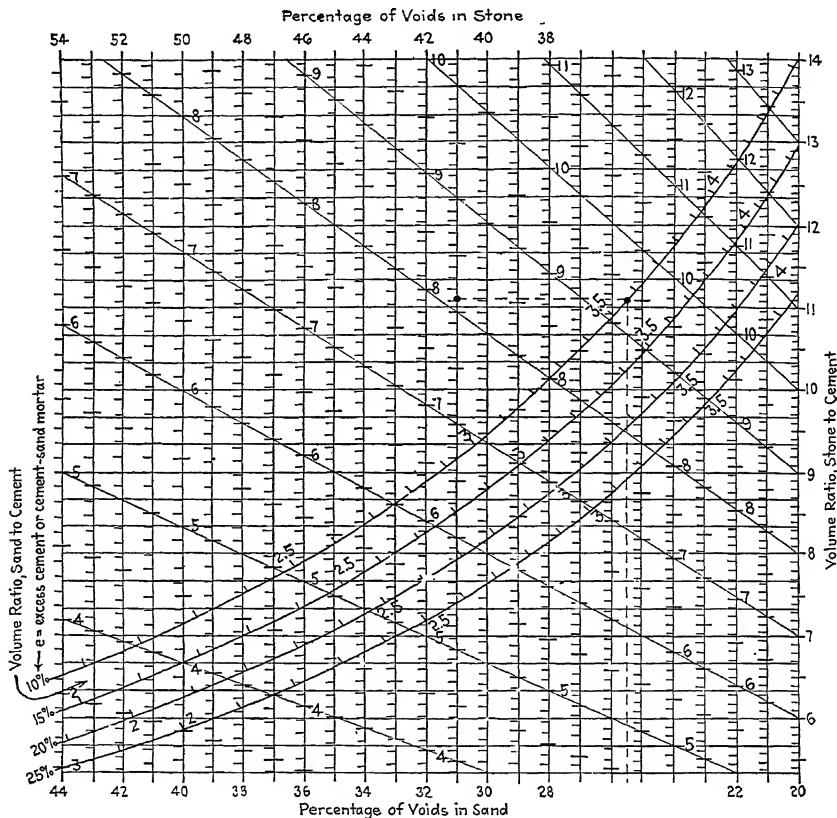
White Portland cement is a variety of Portland requiring raw materials almost free from iron oxide, and fuel almost free from pyrite. Its special value is, of course, architectural.

Blended Cements.—Johnson's "Materials of Construction" (5th ed.) describes several instances of large concrete construction undertaken with blended cement—a mixture, usually of 50 to 55 per cent of Portland cement and 50 to 45 per cent of crushed rock, ground together until 90 per cent passes 200 mesh. The rock in one case was granite, in another pumaceous tufa; it should preferably contain soluble silica. Concrete made with blended cement sets more slowly than the usual Portland-cement concrete, and in the 1:2:4 mixture was reported in one case to have a compressive strength about 20 per cent less, although in poorer mixtures the difference was much reduced.

Puzzolan and Slag Cement.—Puzzolan is a ground mixture of volcanic ash (pozzuolana) and hydrated lime. Slag cement—the American equivalent—replaces the volcanic ash with granulated iron blast-furnace slag; its strength in rich concrete mixtures is less than that of Portland, but in lean mixtures the difference is not so marked.

Concrete (Dana) should be an aggregate of broken stone with voids completely filled by cement-sand mortar, the mortar itself being an aggregate of sand with voids completely filled by cement paste. In practice there should be a slight excess of cement in the mortar and of mortar in the concrete, above the voids requirement. Measure the voids on the job. For broken stone, compute from the volume of water absorbed by a known volume of stone; for sand, pour a pint of dry sand into a quart measure half full of water and compute from the unfilled volume remaining; *e.g.*, if the

water rises to the first gill mark from the top, the sand contains 25 per cent voids. *Fine aggregate*—sand and the screenings of gravel or crushed stone—should all pass $\frac{1}{4}$ in., contain not more than 30 per cent of material through 50 mesh, and be clean and free from soft particles; *coarse aggregate*—gravel and crushed stone—should be larger than $\frac{1}{4}$ in., clean, hard and free from flat and elongated pieces (Committee of



Cement : Sand : Stone Ratios of Concrete Equivalent to Stated Values of Excess of Cement and Cement-Sand Mortar, Over the Volumes Required to Fill Voids in Sand and Stone Respectively.

Example (dotted line): Voids in Sand, 25.5%; in Stone, 41%; Excess = 10%.
Solution: Ratio Sand to Cement, 3.56; Stone to Cement, 8.11

FIG. 27.—Concrete—grading of aggregate. Copyright J. A. Church.

the A.S.C.E.). Both should be graded as to size, and voids can often be reduced by mixing sizes. Dana notes a good 1:4 to 1:6 cement-gravel concrete (without sand), e being 10 per cent, obtained by mixing gravel sizes until voids were reduced to 15 to 20 per cent. *Rubble concrete* contains large clean stones, saturated with water and embedded in the usual concrete mixture. It is strong and saving in cement, but

suitable to massive work only (without reinforcement). *Cinder concrete* replaces stone with hard, clean vitreous clinker, free from ash and unburned coal. It is relatively weak, hence excluded from reinforced work except in floor slabs not over 8 ft. in span. Weight in pounds per cubic foot of stone concrete, 150; of cinder concrete, 110.

Proportions.—Dana recommends the following values of e^1 : for foundations, 10 per cent; for abutments and piers, 15 per cent; for reinforced work, 20 per cent; for thin sections, slabs, and waterproof work, 25 per cent (with excess of water). To convert these specifications into the usual cement-sand-stone ratios, use the chart (Fig. 27). The Committee of the A.S.C.E. recommends the following ratios of cement to total aggregates: for reinforced work, 1:6; for massive masonry or rubble concrete, 1:9 to 1:12, ratio of fine to coarse aggregate to be that producing maximum density.

Mixing and Placing.—Unit of measure, the 94-lb. bag of cement, equivalent to 1 cu. ft. (Committee of A.S.C.E.); the barrel of cement, yielding an average of 3.5 cu. ft. of cement paste (Dana). Good concrete requires measured proportions, thorough mixing, and the right proportion of water—enough to produce a mixture that flows sluggishly and quakes slightly under ramming, except in waterproof work, which requires a little more. Water for mixing should be free from oil, acid, alkali, and organic matter. Concrete should be placed as soon as mixed and rammed briefly; if delivered to place by spout, it should flow in a continuous stream. Segregation of ingredients impairs strength; it may be due to too much water or to intermittent flow in the spout. Frozen concrete is dangerous; if placing must be done in freezing weather, the ingredients are warmed beforehand and slabs, beams, and columns kept warm until set. Massive masonry once placed requires less care. Concrete does not always set as quickly as expected, and should be tested for a distinctive ring under the hammer before forms are removed; frozen concrete may be mistaken for material properly set.

Design of massive concrete (not reinforced) employs the compressive strength of the material to carry direct load and its weight to resist overturning forces; its slight tensile strength is not utilized. Concrete shrinks in setting, and in large masses contraction joints are provided every 25 to 50 ft. They may be tongued and grooved to preserve alignment.

Working stresses for direct compression, expressed as percentages of the ultimate strengths shown in Table 70 (Committee of A.S.C.E.), are for a large mass loaded over not more than half its surface, 35 per cent (applied to the loaded area only); for a pier concentrically loaded, whose length is not more than four diameters, 25 per cent.

TABLE 70.—ULTIMATE COMPRESSIVE STRENGTHS OF CONCRETE

Combined volume of fine and coarse aggregates (measured separately) per unit volume of cement	3	4.5	6	7.5	9
Aggregate	Ultimate compressive strength in pounds per square inch				
Granite, trap rock.	3,300	2,800	2,200	1,800	1,400
Gravel, hard limestone, hard sandstone. . .	3,000	2,500	2,000	1,600	1,300
Soft limestone, soft sandstone.	2,200	1,800	1,500	1,200	1,000
Cinders.	800	700	600	500	400

¹ e signifies the excess of cement and sand over the theoretical requirement.

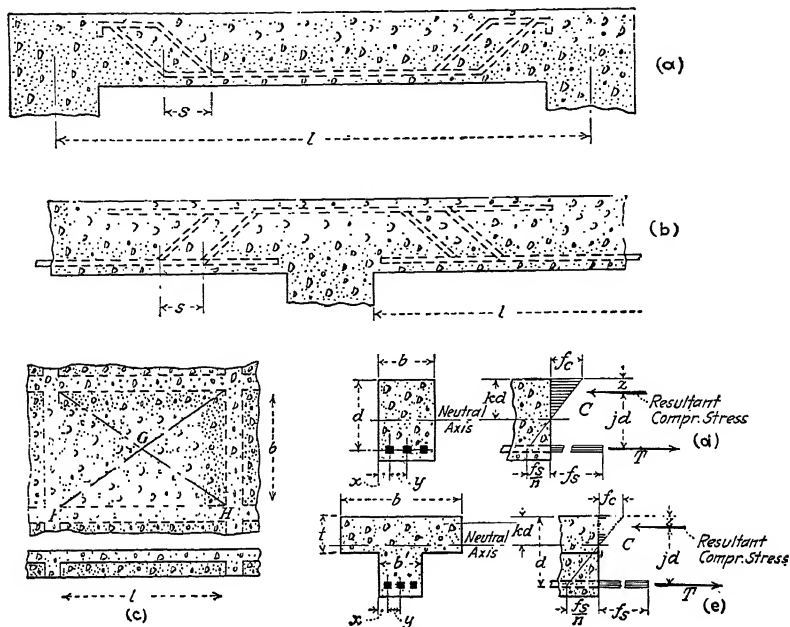


FIG. 28.—Reinforced beams and slabs.

TABLE 71.—BILLET STEEL REINFORCEMENT BARS

Type and grade of bar	Tensile strength minimum or range, pounds per square inch		Minimum per cent elongation in 8 in. ¹	Bend test, sizes			
				Under $\frac{3}{4}$ in.		$\frac{3}{4}$ in. and over	
	Ultimate	Yield point		Arc	d	Arc	d
Plain bars:							
Structural.....	55,000-70,000	33,000	1.40	180°	t	180°	t
Intermediate.....	70,000-85,000	40,000	1.30	180°	$2t$	90°	$2t$
Hard.....	80,000 min.	50,000	1.20	180°	$3t$	90°	$3t$
Deformed bars:							
Structural.....	55,000-70,000	33,000	1.25	180°	t	180°	$2t$
Intermediate.....	70,000-85,000	40,000	1.125	180°	$3t$	90°	$3t$
Hard.....	80,000 min.	50,000	1.00	180°	$4t$	90°	$4t$
Cold-twisted.....	55,000	5 per cent	180°	$2t$	180°	$3t$

¹ Minimum per cent elongation = tabular coefficient \times 1,000,000 \div ultimate strength. For plain and deformed bars, these elongations to be reduced at the rate of 0.25 per cent for each $\frac{1}{32}$ -in. increase of specified thickness or diameter above $\frac{3}{4}$ in., and at the rate of 0.5 per cent for each $\frac{1}{32}$ -in. decrease below $\frac{3}{4}$ in.

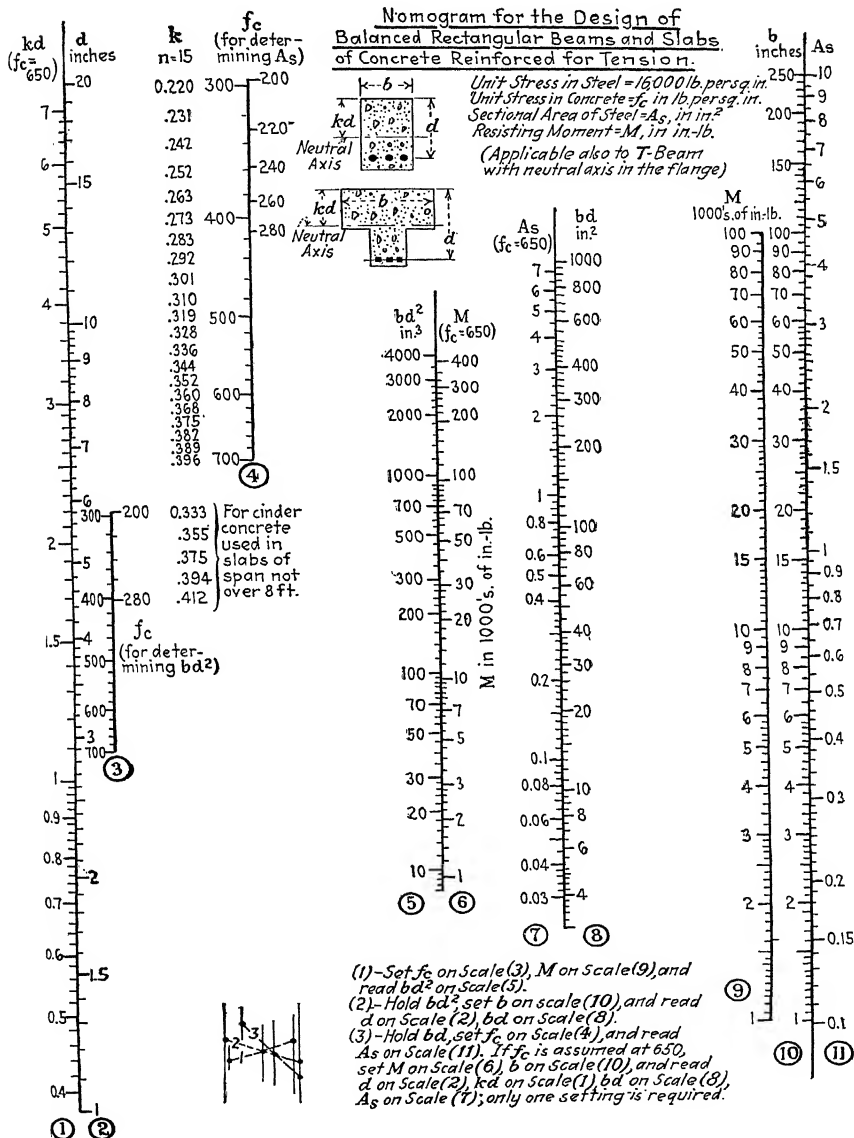


FIG. 29.—Concrete beams and slabs. Copyright J. A. Church.

Design of Reinforced Concrete.—In members subject to bending, concrete is used as the compressive element and embedded steel reinforcement as the tensile. To transmit tensile stress to the steel, reliance is placed largely on its adhesion to the surrounding concrete (*bond*), although reinforcement may be—in the case of cantilever beams must be—anchored to supports. The following treatment is that recommended by the Committee of the A.S.C.E. except as noted.

Reinforcement may consist of plain steel bars, deformed bars (with surfaces ridged or roughened to improve bond), or—for slabs, small beams, and shrinkage reinforcement—steel wire, expanded metal, etc. A.S.T.M. specifications (A-15) for billet steel bars require Bessemer or openhearth steel, graded as structural, intermediate, and hard; maximum phosphorus in Bessemer, 0.10 per cent; in openhearth, 0.05 per cent. Table 71 shows other requirements.

Sizes (Carnegie): of plain square bars, $\frac{3}{16}$ to 2 in.; of rounds, $\frac{1}{32}$ to $1\frac{3}{4}$ in. (both inclusive); advance by sixty-fourths (for cold-twisted bars, see Table 72).

TABLE 72.—COLD-TWISTED BARS

Index	Size, inches	Pounds per foot	Index	Size, inches	Pounds per foot
$\frac{1}{4}$	0.0625	0.213	$1\frac{1}{16}$	0.4727	1.607
$\frac{5}{16}$	0.0977	0.332	$\frac{3}{4}$	0.5625	1.913
$\frac{3}{8}$	0.1406	0.478	$1\frac{3}{16}$	0.6602	2.245
$\frac{7}{16}$	0.1914	0.651	$\frac{7}{8}$	0.7656	2.603
$\frac{1}{2}$	0.2500	0.850	$1\frac{5}{16}$	0.8789	2.988
$\frac{9}{16}$	0.3164	1.076	1	1.0000	3.400
$\frac{5}{8}$	0.3906	1.328	$1\frac{1}{8}$	1.2656	4.303

Reinforcement for shrinkage in large areas of concrete exposed to atmospheric conditions is well distributed near the exposed face in amount not less than one-third of 1 per cent of the gross area. If a joint is necessary, it is made at a point of minimum stress and reinforcement is provided across the joint, either anchored or extended each way until enough bond is secured to develop full tensile strength. Fillets at the junction of members of contrasted size aid in reducing shrinkage cracks.

General Data.—For notation see Table 73. Premises: E_s is constant within working limits; tensile strength of concrete is neglected, as is initial stress in steel due to shrinkage of concrete; adhesion of steel and concrete is perfect, whence in compres-

TABLE 73.—REINFORCED-CONCRETE NOTATION

	Concrete	Steel
Elastic modulus, pounds per square inch.	E_c	E_s
Unit stress, pounds per square inch.....	f_c	f_s
Sectional area, square inch.....	A_s
Total compressive stress, pounds.....	C	
Total tensile stress, pounds.....	T	
Total shear, pounds. . . V , producing stress in steel.....	V'	
Perimeter of one bar, in. . . \circ , of all bars.....	$\Sigma \circ$	
Ratio of A_s to sectional area of member.....	p	
	$n = E_s/E_c$	

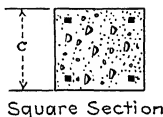
Nomogram for the Design of Reinforced Concrete Columns

A—Longitudinal reinforcement only, representing not less than 1% nor more than 4% of column value, the lateral ties (between longitudinal bars) not less than $\frac{1}{4}$ inch thick, spaced to a maximum of 12 inches or 16 diameters of longitudinal bar—whichever is less. Column length not over 12 diameters of column.
B—Longitudinal reinforcement as above, combined with circular hoops or

Safe Load P
in 1000's. of lb.

spacing of hoop
enclosed diameter, and n
10 enclosed diameters.

In the case of **B** columns,
 c or d is the "enclosed"
dimension (inside hoops, etc.)



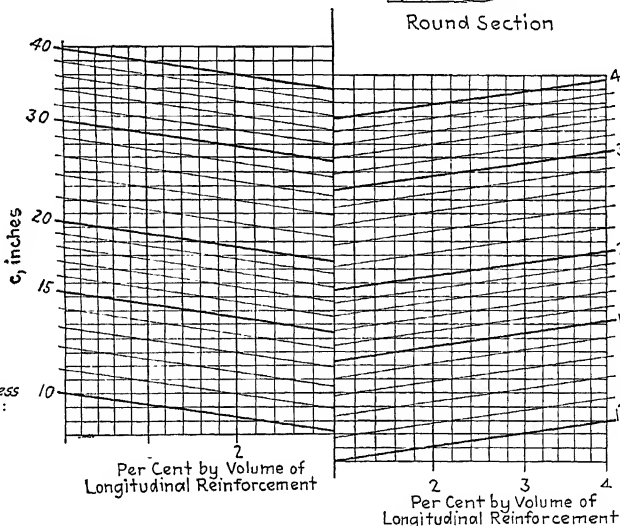
Ultimate
Compressive
Strength of
Concrete,
lb. per sq. in.

B

A

1500
2000
2200
1500
2000
2200

Working Stress
in Concrete:
 $f_c = 650$



Using preferably a transparent straight-edge,
set the ultimate strength of concrete on either
Scale (1) or Scale (2) (depending on type of
reinforcement), and load on Scale (4).
The horizontal line through the intersection of
the edge with Scale (3) cuts the vertical line
representing per cent of longitudinal reinforcement
at the required point—i.e. the point at which
the inclined graduations read the required
value of c or d .

Least over-all
thickness or
diameter, 12 in

$n = 15$

④



FIG. 30.—Reinforced concrete columns. Copyright J. A. Church

sion they are stressed in the ratio n . Value of n is taken at 15, except in the calculation of kd (Fig. 28), of resisting moment in beams and compression of concrete in columns, to which Table 74 applies, and in the calculation of beam deflections, for which $n = 8$.

TABLE 74.—VALUES OF n

ULTIMATE COMPRESSIVE STRENGTH OF CONCRETE, POUNDS PER SQUARE INCH	
Not over 800.....	40
800–2,200.....	15
2,200–2,900.....	12
Over 2,900.....	10

Reinforced-concrete beams and slabs, if combined in several panels—the slabs supported along four sides—are commonly designed as continuous over supports; but in such case ample stability must be provided in foundations, as slight changes in alignment produce serious adjustments in the calculated stresses. Beams are typically reinforced by longitudinal bars embedded near the tension face; certain types require special web reinforcement, for which see *Trans. A.S.C.E.*, Vol. 81, p. 1101; only the simpler and more general types are treated here.

Shear.—Vertical shear and horizontal stresses due to bending produce resultant diagonal stresses in the concrete, and insofar as diagonal tension is involved, its practicable intensity is limited by the slight tensile strength of the material. Diagonal tension, though not directly proportional to, is best measured by the vertical shear itself. The simplest method of reinforcing against it is to bend the longitudinal bars diagonally through the web (Figs. 28a, b). In a continuous beam, reinforcement for negative moments at the supports (Table 2) is often combined with diagonal reinforcement by bending up half the bars in each span and extending them beyond the supports far enough to secure bond, the overlap of bars from two adjacent spans furnishing the full steel area over each support (Fig. 28b). The effect of a bent bar in resisting diagonal tension is considered to extend horizontally no farther than $\frac{3}{4}d$ from the bend, and (in constrained beams) not beyond the point of zero moment; hence s is $\frac{3}{4}d$ or less, and bends are made at distances of about $l/5$ from supports. For bars bent at 20 to 45 deg., diagonal tension in one bar = $\frac{3}{4}(V's/jd) = \frac{1}{2}(Vs/jd)$, but the total shear for which provision is thus made by bent bars must not exceed a value equal to 4.5 per cent of the ultimate compressive strength of the concrete (Table 70) applied to effective beam section bd . The total shear not carried by bent bars must be allotted to the concrete itself, and unit shear in concrete ($= V/bjd$) must not exceed 2 per cent of its compressive strength.

Bond stress per unit area of bar = $V/jd\sum o$. Allowable bond stress, expressed as a percentage of ultimate compressive strength of concrete (Table 70): plain bars, 4 per cent; best types of deformed bar, 5 per cent; drawn wire, 2 per cent.

External moment M (inch-pound) for span l (inches) and uniform load of w lb. per in.: in floor slabs of more than two spans continuous over supports, $\pm(wl^2/12)$; of two spans only, $\pm(wl^2/10)$; in end spans of continuous beams, $wl^2/10$ at center, $-(wl^2/10)$ at inward support, and $-(wl^2/12)$ at outward support; in interior spans, $\pm(wl^2/12)$; in simple beams, $wl^2/8$ at center. In each case w is combined dead and live load, the latter properly increased for impact; concentrated loads require individual treatment. Span l is measured for simple beams as in Fig. 28a; for continuous or constrained beams, as in Fig. 28b. In a slab supported along four sides (Fig. 28c) with uniform load W , the transverse reinforcement is designed for moment due to $W(l/b - \frac{1}{2})$, up to a maximum of W ; the longitudinal reinforcement is allotted the remainder. Load

transmitted by such a slab to any supporting beam is regarded as distributed according to ordinates from FH to FGH [Fig. 28c (Ketchum)].

Rectangular Beams, Tensile Reinforcement Only (Fig. 28d).—For equilibrium $C = T$ and $M = C \cdot jd = \frac{1}{2}f_c \cdot kd \cdot b \cdot jd = Tjd = f_s \cdot A_s \cdot jd$, whence $A_s/bd = \frac{1}{2}k \times f_c/f_s = p$. Lever arm $jd = d - kd/3$ and $j = 1 - k/3$. Unit strain (tensile or compressive) is assumed to be a function of distance from the neutral axis, or

$$f_s/E_s \div f_c/E_c = (d - kd) \div kd, \text{ whence } f_s/n \div f_c = (1 - k) \div k \text{ or } k = nf_c/(f_s + nf_c)$$

and p (for "balanced" reinforcement) $= nf_c^2/[2f_s(f_s + nf_c)]$; thus f_s and f_c determine k , j , and p , whence M fixes bd (and therefore A_s); and b , known or assumed, determines d ; for rapid solution use Fig. 29. If more steel is used than required by the foregoing value of p , the strength of the beam is limited by that of the concrete; if less, by that of the steel. Maximum allowable stress: $f_s = 16,000$ lb. per sq. in.; $f_c = 32.5$ per cent of the ultimate compressive strength of the concrete (Table 70), except near supports of continuous beams, where f_c may be 37 per cent.

T Beams (Fig. 28e).—In beam-and-slab construction, each interior beam may be regarded as including portions of the adjacent slabs, the combined T section being designed as a unit. If its proportions are so chosen that its neutral axis lies in the flange, use the formulas or nomogram for rectangular beams (wherein b is the full flange width); t must then exceed kd , and the relative proportions of slab and beam control design. This condition represents conservative practice, suited to the heavy duty required of most metallurgical structures; for the treatment of a T beam whose neutral axis lies in the stem, see the reference. The T section is sometimes used in other than beam-and-slab construction, to secure required compression area. The A.S.C.E. Committee recommends that b shall not exceed $l/4$, and that the overhanging width of flange shall not exceed $6t$ on either side. Dana limits the overhang to t on either side, for security and simplicity of design.

Spacing of Reinforcement (Figs. 28d, e).—Minimum x = two thicknesses, minimum y = three thicknesses of bar; spacing between bars and lower (or upper) face of beam, 1 to $1\frac{1}{2}$ in. in the clear, depending on bar size. If two layers of bars are used, they are spaced at least 1 in. apart in the clear; more than two layers are rare.

Reinforced-concrete columns are reinforced (1) by longitudinal bars, (2) by these combined with bands, hoops, or spirals, (3) by structural shapes. Effective area in the two latter cases is the area of the circle enclosing hoops or spirals, or of the polygon enclosing the shapes. The ratio of unsupported length to least width is 15, maximum. The following treatment applies to columns whose over-all width or diameter is at least 12 in.; it is assumed that compressive stress is distributed between longitudinal bars and concrete in the ratio n ; hoops, bands, and spirals increase the toughness of a column, but are assigned no direct share in sustaining load.

Working Stresses.—For steel, $f_s = 16,000$ lb. per sq. in.; for concrete, f_c (expressed in percentage of ultimate compressive strength, Table 70), is as follows: For a column whose length does not exceed 12 diameters, with longitudinal reinforcement representing not less than 1 per cent or more than 4 per cent of the column volume, and with lateral ties (between bars) not less than $\frac{1}{4}$ in. in diameter spaced to a maximum of 12 in. or 16 diameters of longitudinal bar (whichever is less), $f_c = 22.5$ per cent. For a column (with hoops or spirals) whose length does not exceed 10 enclosed diameters, with longitudinal reinforcement representing not less than 1 per cent or more than 4 per cent of the enclosed volume, and with circular hoops or spirals representing not less than 1 per cent of that volume; $f_c = 34.8$ per cent. In the latter case the maximum vertical spacing of hoops or spirals is limited to $\frac{1}{6}$ (preferably $\frac{1}{10}$) the enclosed diameter, and not more than 2.5 in. Hoops must be securely joined at the ends.

Safe load P (pounds) = $f_c(A_c + nA_s) = f_cA[1 + (n - 1)p]$, wherein A_c is the sectional area of concrete and A the total net area of column (square inch), other notation as before. Stresses, $f_s = nf_c$. For rapid solution use Fig. 30. Bending stresses due to eccentric loads require increase in the column section until maximum stresses are within the foregoing limits. Unequal spans of a continuous beam apply a moment to the intervening column measured by the difference between their negative moments over the column head (see Combined Stresses). Tension in longitudinal bars requires good connection at their ends.

Stone.—Table 75, listing average unit weights of stone, is condensed from a table in Johnson's "Materials of Construction," 5th ed. (after Eckel). Table 76 shows strengths according to Kidder.

TABLE 75.—UNIT WEIGHTS OF STONE

Type	Pounds per cubic foot		
	Maximum	Minimum	Average
Trap rock.....	189	175	184
Granite.....	177	165	168
Sandstone.....	172	132	157
Limestone.....	172	155	165
Marble.....	179	171	175
Slate.....	177	172	174

Brick.—This does not include the refractory bricks, for which see Chap. XI. *Building brick* (A.S.T.M., C-21) is classified as vitrified, hard, medium, and soft; standard size, $2\frac{1}{4} \times 3\frac{3}{4} \times 8$ in. In the absorption test, dry brick is immersed, in water at 60 to 80°F.; the water is brought to boiling within 1 hr., boiled 5 hr.

TABLE 76.—CRUSHING RESISTANCE OF STONE (PRESSURE AT RIGHT ANGLES TO BED)

	POUNDS PER SQ. IN.
Granites:	
Blue, Fox Island, Maine.....	14,875
Gray, Vinal Haven, Maine.....	13,000–18,000
Westerly, R.I.....	15,000
Rockport and Quincy, Mass.....	17,750
Milford, Conn.....	22,600
Staten Island, N.Y.....	22,250
East St. Cloud, Minn.....	28,000
Gunnison, Colo.....	13,000
Red, Platte Cañon, Colo.....	14,600
Limestones:	
Glens Falls, N.Y.....	11,475
Joliet, Ill.....	12,775
Bedford, Ind.....	6,000–10,000
Salem, Ind.....	8,625
Red Wing, Minn.....	23,000
Stillwater, Minn.....	10,750

TABLE 76.—CRUSHING RESISTANCE OF STONE (PRESSURE AT RIGHT ANGLES TO BED)
(Continued)

	POUNDS PER Sq. IN.
Sandstones:	
Dorchester, N.B. (brown).....	9,150
Mary's Point, N.B. (fine grain, dark brown).....	7,700
Connecticut brown stone, ¹ on bed.....	7,000-13,000
Longmeadow, Mass. (reddish brown).....	7,000-14,000
Longmeadow, Mass. (average, for good quality).....	12,000
Little Falls, N.Y.....	9,850
Medina, N.Y.....	17,000
Potsdam, N.Y. (red).....	18,000-42,000
Cleveland, Ohio.....	6,800
North Amherst, Ohio.....	6,212
Berea, Ohio.....	8,000-10,000
Hummelstown, Pa.....	12,810
Fond du Lac, Minn.....	8,750
Fond du Lac, Wis.....	6,237
Manitou, Colo. (light red).....	6,000-11,000
St. Vrain, Colo. (hard laminated).....	11,505
Marbles:	
Lee, Mass.....	22,900
Rutland, Vt.....	10,746
Montgomery County, Pa.....	10,000
Colton, Cal.....	17,783
Italy.....	12,156
Flagging:	
North River, N.Y.....	13,425

¹ This stone should not be set on edge.

and cooled to 60 to 80°F. The gain in weight of the brick, after removal of surface water, indicates the percentage of absorption. The compression test is made on a half brick, placed on edge; to secure a true bearing, each face under compression is first

TABLE 77.—BUILDING AND CLAY SEWER BRICK

Class	Absorption limits, per cent		Compressive strength, pounds per square inch		Modulus of rupture, pounds per square inch	
	Mean of five tests	Indi- vidual maxi- mum	Mean of five tests	Indi- vidual mini- mum	Mean of five tests	Indi- vidual mini- mum
Building brick:						
Vitrified.....	5 or less	6.0	5,000 or over	4,000	1,200 or over	800
Hard.....	5-12	15.0	3,500 or over	2,500	600 or over	400
Medium.....	12-20	24.0	2,000 or over	1,500	450 or over	300
Soft.....	20 or over	No limit	1,000 or over	800	300 or over	200
Clay sewer brick:						
Class A vitrified.....	3 or less	5,000 or over	1,200 or over	800
Class B vitrified.....	5 or less	6.0	5,000 or over	4,000	1,200 or over	800
Hard.....	5-10	12.0	3,500 or over	2,500	600 or over	400
Medium.....	10-15	17.0	2,000 or over	1,500	450 or over	300

shellacked, then bedded in plaster of Paris molded to a plane surface, and applied as thinly as possible. Depressions in a recessed or paneled brick are filled with neat Portland cement mortar and held for 24 hr. before test. In the transverse loading test, a dried brick is placed with its side resting on knife-edges 7 in. apart and center loaded to rupture; the knife-edges are slightly curved lengthwise. Modulus of rupture = $3Wl/2bd^2$, wherein l = span, b = breadth, d = depth of brick (all in inches), and W = breaking load (pounds). Each of these tests is made on at least five specimens; requirements are shown in Table 77. Values for the compressive strength of brick piers, representing averages of results obtained by the National Bureau of Standards, are shown in Table 78.

Clay sewer brick (A.S.T.M., C-32) is classified as class A vitrified, class B vitrified and hard (for sewage velocities of 18, 12, and 8 ft. per sec., respectively), and medium (for low velocities, in storm sewers, for which permeability is not important). Limiting sizes, maximum and minimum, are $2\frac{1}{4} \times 3\frac{3}{4} \times 8$ and $2\frac{1}{2} \times 4 \times 8\frac{1}{2}$ in., except that class A vitrified may be $3 \times 4 \times 8\frac{1}{2}$ (upper limit $3\frac{1}{2} \times 4 \times 8\frac{1}{2}$). Sewer brick is subject to the same tests as building brick, and requirements are shown in Table 77.

TABLE 78.—AVERAGE ULTIMATE COMPRESSIVE STRENGTHS OF BUILDING-BRICK PIERS

Brick	Mortar		months	Average ultimate, pounds per square inch
	Cement	Cement to sand		
Common	Lime	1:6		171
Hard. . . .	Lime	1:6		907
Vitrified.	Lime	1:6		1,360
Hard. . . .	(a)	1:3		1,460
Vitrified.	(a)	1:3		2,900
Common	Portland	1:3		570
Hard. . . .	Portland	1:3		1,650
Vitrified.	Portland	1:3		2,780

(a) 15 lime: 85 portland.

Paving brick (A.S.T.M., C-7), aside from visual inspection, is subjected only to the rattler test, in which sample bricks and cast-iron balls are tumbled together for an hour in a rotating drum; the resistance of the brick to abrasion is gauged by its loss in weight during the test. An average loss of 22 per cent indicates use under heavy traffic conditions; 24 per cent, medium traffic conditions; and 26 per cent, light traffic conditions. The construction of the rattler, etc., is specified in detail.

Hollow tile, among its many applications, is sometimes used in the construction of reinforced-concrete slabs; tiles are set into the concrete on the lower (tension) side between reinforcing bars, and save the equivalent volume of cement. Their use in partitions and walls is familiar. They are, of course, effective in reducing dead load.

Clay sewer pipe (A.S.T.M., C-13) is subjected to a crushing load (at a rate of about 2000 lb. per min.) applied longitudinally from end to end of the piece, excluding socket; load may be applied through knife-edges, which are brought to a true bearing by fillets of plaster of paris and sand, or by three-edge wooden bearings (one above and two, 1 in. apart, below) or by sand bearings. In each case the bearing is designed

to compensate for surface inequalities in the pipe. The hydrostatic test requires freedom from leakage under successive pressures of 5 lb. for 5 min., 10 lb. for 10 min., and 15 lb. for 15 min., all per square inch. In the absorption test, the specimen (broken piece) is dried to constant weight, covered with water and boiled for 5 hr., then cooled in the water. Table 79 shows minimum requirements and sizes. This pipe is not usually vitrified.

TABLE 79.—CLAY SEWER PIPE

Inside diameter, inches	Laying length, feet	Inside diameter at mouth of socket, inches	Depth of socket, inches	Thickness of barrel, inches	Average ultimate compressive strength, pounds per square inch		Maximum absorption, per cent
					Knife-edge and three-edge bearings	Sand bearings	
4	2	6	1½	⅞	1,000	1,430	8
6	2	8½	2	⅞	1,000	1,430	8
8	2, 2½, 3	10¾	2½	¾	1,000	1,430	8
10	2, 2½, 3	13	2½	¾	1,100	1,570	8
12	2, 2½, 3	15¼	2½	1	1,200	1,710	8
15	2, 2½, 3	18¾	2½	1¼	1,370	1,960	8
18	2, 2½, 3	22¼	3	1½	1,540	2,200	8
21	2, 2½, 3	26	3	1¾	1,810	2,590	8
24	2, 2½, 3	29½	3	2	2,150	3,070	8
27	2½, 3	33¼	3½	2¼	2,360	3,370	8
30	2½, 3	37	3½	2½	2,580	3,690	8
33	2½, 3	40¼	4	2⅝	2,750	3,930	8
36	2½, 3	44	4	2¾	3,080	4,400	8
39	2½, 3	47¼	4	2⅞	3,300	4,710	8
42	2½, 3	51	4	3	3,520	5,030	8

Minimum taper of socket, 1:20. Thickness of socket ¼ in. from its outer end, not less than 75 per cent of thickness of barrel.

Cement-concrete sewer pipe (A.S.T.M., C-14) is tested by the methods described for clay sewer pipe, and the requirements are the same (Table 79). Dimensions differ slightly, however, and are shown in Table 80.

TABLE 80.—DIMENSIONS OF CEMENT-CONCRETE SEWER PIPE

Inside diameter, inches	Laying length, feet	Inside diameter at mouth of socket, inches	Depth of socket, inches	Thickness of barrel, inches	Inside diameter, inches	Laying length, feet	Inside diameter at mouth of socket, inches	Depth of socket, inches	Thickness of barrel, inches
4	2, 2½	6	1½	⅞	24	2, 2½, 3	29½	3	2½
6	2, 2½	8½	2	⅞	27	2½, 3	33¼	3½	2¾
8	2, 2½, 3	10¾	2½	¾	30	2½, 3	37	3½	2⅝
10	2, 2½, 3	13	2½	¾	33	2½, 3	40¼	4	2¾
12	2, 2½, 3	15¼	2½	1	36	2½, 3	44	4	3
15	2, 2½, 3	18¾	2½	1¼	39	2½, 3	47¼	4	3¼
18	2, 2½, 3	22¼	3	1½	42	2½, 3	51	4	3½
21	2, 2½, 3	26	3	1¾					

Minimum taper of socket, 1:20. Thickness of socket ¼ in. from its outer end, not less than 75 per cent of thickness of barrel.

TIMBER

General.—Tree growth in the spring is rapid, and “spring wood” is not so dense and strong as the more slowly formed “summer wood”; similarly, narrow annual rings imply slow growth, strength, and density. As a tree develops, the wood cells at its heart no longer function, but provide a strong central support for the active sapwood nearer the bark, and furnish the best material in the log. Timber is not a homogeneous material; it usually fails at some point of local weakness, and its ultimate strength varies with the character of the piece more markedly than that of any other common structural material. It fails under a long-continued load much below the ultimate as determined in the testing laboratory, and its strength (especially in compression) is adversely affected by moisture, which itself varies not only from piece to piece, but also in the single piece after erection. Painting and preservative methods exclude moisture, and safe working stresses and good specification guard against failure from local defects.

Classification (A.S.T.M., D-9, condensed).—*Southern yellow pine* (longleaf, shortleaf, loblolly, Cuban, and pond pines) is the species found from Virginia to Texas. It is “dense” if it shows at least one-third summer wood and an average of six or more annual rings per inch on either end, or at least one-half summer wood though wide ringed; “sound,” if of good quality but unable to qualify as dense. *Norway pine* includes what is known as “red pine.” *White pine* is timber of this name from Maine, Michigan, Wisconsin, and Minnesota. *Idaho white pine* is the variety from western Montana, northern Idaho, and eastern Washington. *Western pine* (western yellow, Ponderosa, California white, and western white pine) is timber sold as white pine from Colorado, New Mexico, Arizona, California, Oregon, and Washington. *Douglas fir* includes yellow fir, red fir, western fir, Washington fir, Oregon or Puget Sound fir or pine, Northwest, and West Coast fir. *Hemlock* comes from Minnesota and states eastward, *western hemlock* from the Pacific coast. *Spruce* comes from Minnesota and states eastward, *western spruce* from the Pacific coast. *Tamarack* (eastern tamarack) comes from Minnesota and states eastward. *Western larch* is larch or tamarack from the Rocky Mountain and Pacific coast regions. *Redwood* is the California wood of this name.

Defects (A.S.T.M., D-9, condensed).—*Knots* are not necessarily defects, although limited in number and size by good specification. A *sound knot* is solid across its face, as hard as the enclosing wood, and firmly held in place by growth or position; if not over $\frac{1}{2}$ in. in diameter it is a *pin knot*, if not over $1\frac{1}{2}$ in. a *standard knot*, if over $1\frac{1}{2}$ in. a *large knot*; if it shows a pith hole at center not over $\frac{1}{4}$ in. in diameter, it is a *pith knot*. An *encased knot* is one whose rings are not intergrown and homogeneous with those of the surrounding piece; if the rings are partially intergrown and the knot is held in place by growth or position, it is a *sound knot*; if intergrowth is complete on one face, a *watertight knot*. A *loose knot* is one not held firmly in place by growth or position. A *rotten knot* is one not so hard as the enclosing wood. Sawing to a round or oval section produces a *round knot*, sawing lengthwise a *spike knot*.

A *pitch pocket* is an opening between the grain, containing pitch or bark; it is *small* if it is not over $\frac{1}{8}$ in. wide, *standard* if not over $\frac{3}{8}$ in. wide nor over 3 in. long, *large* if either dimension exceeds that of the standard. *Pitch streak* is a well-defined accumulation of pitch; where no well-defined streak appears, or where coarse-grained fiber between the grain (spring wood) is not saturated with pitch, no defect is involved.

Wane is bark, or lack of wood from any cause, on edges of timbers. *Shakes* are splits or checks—a *ring shake* separates annual rings, a *through shake* extends between opposite faces of the pieces. *Rot*, *dote*, and *red heart* are forms of decay. Any decay, in the form of a dark-red discoloration absent from the sound wood, or of white or red rotten spots, is a defect.

TABLE 81.—STRENGTH OF STRUCTURAL TIMBER

All stresses, pounds per square inch	Bending			Shearing			Compression			
	Extreme fiber stress		Elastic modulus	Parallel to grain		Longitudinal stress in beams	Perpendicular to grain		Working stress, columns	
	Ultimate	Working		Ultimate	Working		Elastic limit	Working stress		
Timber	Ultimate	Working		Ultimate	Working	Ultimate	Working	Ultimate	Working	
Longleaf pine.....	6,500	1,300	1,610,000	720	180	300	120	3,800	975	1,300
Shortleaf pine.....	5,600	1,100	1,480,000	710	170	330	130	3,400	825	1,100
Douglas fir.....	6,100	1,200	1,510,000	690	170	270	110	3,600	900	1,200
Norway pine.....	4,200	800	1,190,000	590	130	250	100	2,600	600	800
Western hemlock.....	5,800	1,100	1,480,000	630	160	270	100	3,500	900	1,200
Spruce.....	4,800	1,000	1,310,000	600	130	170	70	3,200	825	1,100
White pine.....	4,400	900	1,130,000	400	100	180	70	3,000	750	1,000
Tamarack.....	4,600	900	1,220,000	670	170	260	100	3,200	755	1,000
Redwood.....	5,000	900	800,000	500	80	3,300	675	900
Bald cypress.....	4,800	900	1,150,000	500	120	3,900	825	1,100
Red cedar.....	4,200	800	800,000	2,800	675	900
White oak.....	5,700	1,100	1,150,000	840	210	270	110	3,500	975	1,300

Columns: l = length, d = least side in the same dimension.1 For columns in which l/d is over 15, the safe load per square inch = $P(1 - l/60d)$.

Specification (Ketchum).—For structures carrying live load, preferably longleaf yellow pine, Douglas fir, white oak, white pine, or western hemlock. For other purposes, shortleaf yellow pine, spruce, or equivalent good timber. All timber should be cut within 18 months of delivery, from sound live trees, and be close grained and solid, and free from defects (injurious ring shakes and crooked grain, unsound knots, knots in groups, decay, large pitch pockets and all other defects that impair strength).

Sizes.—Rough sizes for flooring and boards up to $1\frac{1}{2}$ in. thick shall be not over $\frac{1}{16}$ in. scant from specified size; planks and timbers, $1\frac{3}{4}$ to $5\frac{3}{4}$ in. thick not over $\frac{1}{8}$ in. scant; timbers, 6 in. thick or more not over $\frac{1}{4}$ in. scant. Dressed timber more than $1\frac{1}{2}$ in. is allowed an additional $\frac{1}{8}$ in. scant for each face planed.

Dimension timber, when used for beams, stringers, caps, posts, and sills, must show a minimum of 75 per cent heart on each of the four faces at every point in its length; no loose knots, or knots larger than $2\frac{1}{2}$ in., or one-fourth the width of face, are permitted; knots must not be in groups, and no knot may be nearer the edge of the piece than one-fourth the width of face. When used for other purposes, the timber is allowed 1 in. wane on one edge or $\frac{1}{2}$ in. on two edges, and ring shakes not over one-eighth the length of the piece.

Timber for carrying heavy loads must show a minimum of 90 per cent heart on each of the four faces at every point in its length; must be out of wind and free from shakes, splits, pitch pockets over $\frac{1}{2}$ in. wide or 3 in. long, knots larger than $1\frac{1}{2}$ in. in diameter or closer together than one in any four linear feet on any face (except that knots 1 in. in diameter or less are allowed one in every 3 ft.).

Planks and scantlings, when used for floors, rafters, purlins, girts, struts, and bracing, must be square edged, show one face all heart and a minimum of 75 per cent heart on the remaining faces (measured at any point in its length), and free from loose knots and "large" sound knots. When used for other purposes, knots may be 2 in. in diameter, and wane covering 5 per cent of the surface is permitted in 5 per cent of the total number of pieces of any one size.

Piles are preferably of longleaf yellow pine, white oak, Douglas fir, cedar, or cypress. They must be cut from sound, live trees above ground, swell and taper evenly from butt to tip; be straight, close grained, and solid; be free from defects, such as injurious ring shakes, large and unsound or loose knots, decay, and other defects impairing strength; and be cut square at the ends and stripped of bark. A line from butt to tip must lie within the body of the pile, and short bends are not allowed. Diameters: near butt, 14 in. minimum, 18 in. maximum; at tip, minimum for piles less than 30 ft. long, 8 in., over 30 ft. long, 6 in.

Flooring.—Preferably of yellow pine or maple, usually in lengths of 4 to 16 ft. and width of $2\frac{1}{2}$ in. (dressed). Select flooring must be edge grained, kiln dried, matched, tongued and grooved, free from planer marks, edge splinters, grain slivers, etc.; and must show one face all heart and be free from knots, shakes, sap, and pitch pockets. Common flooring meets the same requirements, except that one solid knot 1 in. in diameter or less is allowed, together with unimportant pitch streaks and sap stains.

Ceiling is graded like flooring, but must also be double beaded and very carefully dressed.

Shingles are of pine, cedar, or cypress. Dimensions: 16 to 18 in. long, 4 to 6 in. wide, $\frac{1}{16}$ in. thick at tip, and $\frac{3}{16}$ to $\frac{1}{2}$ in. at butt. The first 10 in. of length at the butt end must be free from sap, knots, and other defects.

Strength.—Table 81 contains values recommended by the American Railway Engineers Association. The tabular working stresses apply to important structures under severe loading conditions, and for ordinary buildings may be increased 50 per cent. In computing deflections under long-continued bending loads, use one-half the tabular values for elastic modulus. Table 82 contains values for the strength

TABLE 82.—TENSILE STRENGTH OF TIMBER

Kind of timber	Pounds per square inch			
	Ultimate		Working stress	
	With grain	Across grain	With grain	Across grain
White oak.....	12,000	,000	1,200	200
White pine.....	7,000	500	700	50
Southern longleaf or Georgia yellow pine....	12,000	600	1,200	60
Douglas fir.....	8,000	800
Shortleaf yellow pine.....	9,000	500	900	50
Red pine (Norway pine).....	8,000	500	800	50
Spruce and eastern fir.....	8,000	500	800	50
Hemlock.....	6,000	600
Cypress.....	6,000	600
Cedar.....	7,000	700
Chestnut.....	8,500	850
California redwood.....	7,000	700
California spruce.....

of timber in tension. Its chief value is comparative, because timber usually fails (in shear, etc.) at points of attachment before its full tensile strength has been developed. Table 83 shows approximate weights of timber (Johnson's "Materials of Construction," fifth edition).

TABLE 83.—APPROXIMATE WEIGHT OF TIMBER

	Condition	Specific gravity	Pounds per	
			Cubic feet	M board feet
Southern pine, tamarack (best).....	Kiln dried	0.6-0.7	36-42	3,200
Southern pine, pitch pine, tamarack, Douglas fir, western hemlock, maple.	Kiln dried	0.5-0.6	30-36	2,700
Norway pine, red cedar, cypress, hemlock, redwood, chestnut.....	Kiln dried	0.4-0.5	24-30	2,200
White pine, spruce.....	Kiln dried	0.3-0.4	18-24	1,800
Longleaf pine, rough boards.....	Seasoned	3,500
Longleaf pine, dressed.....	Seasoned	3,000

Piles (Ketchum) are driven to firm bearing, or until an average penetration of $\frac{1}{2}$ in. per blow for six blows requires a 2000-lb. hammer falling freely 20 ft. (or equivalent impact)—these figures apply to firm ground. In soft ground, safe load on the pile (tons) = $2Wh/(s + n)$, wherein W is the weight of hammer in tons, h the free fall of hammer in feet, and s the average penetration in inches for the last six blows; n = unity for a gravity and one-tenth for a steam hammer. Total penetration in hard material, at least 10 ft.; in soft material and loam, at least 15 ft.

Preservation.—A rough-and-ready method of fireproofing is to coat with gunite, first covering the timber surface with coarse wire mesh. Preservative methods are designed to prevent lodgment and growth of the fungi that cause decay. Their effectiveness depends upon their success in excluding moisture (which favors fungous growth), upon antiseptic power, and upon depth of penetration of the wood. The following notes are taken from Johnson's "Materials of Construction," fifth edition.

Surface treatment is the cheapest but least effective method. Creosote, paint, oil, and whitewash may be applied with the brush, or the timber may be immersed in preservative and allowed to soak. Charcoal is immune to fungous attack, and timber is occasionally protected by surface charring, but the method is not very effective and impairs strength.

Nonpressure processes require less costly equipment than the pressure methods and effect considerable penetration. They include (1) impregnation with a 1 per cent solution of bichloride of mercury, whose toxic qualities are a serious disadvantage (kyanizing); and (2) heating in a bath of creosote, zinc chloride solution, or crude oil to a temperature just above 100°C.; moisture in the wood cells is converted to steam and on cooling condenses sufficiently to draw in preservative to a considerable depth (open-tank method).

Pressure processes are the most effective (for hemlock, Douglas fir, and redwood the only successful) methods. In "boiling," the timber is soaked in creosote oil and its moisture expelled by heating above 100°C., then impregnated with creosote under pressure. The Bethell process employs coal-tar creosote, also forced into the wood by pressure. It involves a consumption of about 10 to 20 lb. of creosote per cubic foot. Burnettizing uses a 2 to 5 per cent aqueous solution of zinc chloride in the same way, involving a consumption of about 0.5 lb. per cu. ft. of wood. The water solubility of the medium is a drawback in damp locations. Saturation under pressure, followed by vacuum removal of excess preservative, reduces consumption of the preserving medium. In the Rueping process, timber is first put under air pressure, then impregnated with creosote under pressure, after which the compressed air in the wood cells forces out excess preservative (consumption, 5 to 7 lb. per cu. ft. of wood). The Lowry process saturates with creosote under pressure and removes the excess by vacuum. The Card process first subjects a timber to vacuum (to assist penetration), then to immersion under pressure in a mixture of 3 to 5 per cent solution of zinc chloride with 15 to 20 per cent creosote, finally to vacuum (to remove excess preservative), the admixture of zinc chloride making a cheaper preservative. The last three processes secure maximum penetration with comparatively low consumption of the medium.

HEAT AND CORROSION PROBLEMS

In this subdivision materials will be treated from the standpoint of resistance to heat and chemical corrosion. Much of the information quoted in the following pages deals with materials of very special application or of comparatively few makers; or it represents strength testing at high temperature, with a margin of uncertainty that no standard method has yet defined. It, therefore, has not been confirmed by the long experience of the profession, and cannot claim the same authority as the rigorously

tested standards of strength accepted for the common structural materials at ordinary temperature. When a new problem is attacked in the field of resistance to heat or corrosion, consultation with the manufacturers and field test are the first principles of design.

Survey.—Metallurgical design faces a range of working conditions extending from the handling of cold corrosives to the application of flame temperatures. Variations in temperature affect rates of chemical reaction so profoundly that corrosion resistance of some sort is a requisite throughout the range. Between these extremes there are numberless gradations, each a problem in itself, for which very little exists in the way of precedent; and in a great many cases the method of trial and error is the only resource. In such instances the data collected here can do little more than serve as a basis for planning field tests.

Materials for Smelting Temperatures.—At ordinary temperature, structural design can utilize the tensile strength of metals. At smelting temperatures the metals are either seriously weakened or pass to the liquid phase and give place to certain earthy materials (refractories) whose only useful strength is compressive. This fundamental principle, for instance, has fixed the type of the reverberatory furnace, of which the tension members are well insulated from the flame. To some extent, metals have been made to withstand smelting temperatures by the device of jacketing, but the fire sheet of a jacket is subject to such unequal expansion and internal stress that the device itself is not a means of providing members designed for strength alone. In some range below smelting temperatures, therefore, occur the structural limits of usefulness of the metals, and during the last two decades there has been a very general effort to push these limits upward by the development of heat-resistant alloys. The results have already solved many problems and hold some degree of promise for others, but they have not yet brought tensile properties within the scope of smelting temperatures.

Materials for Intermediate Temperatures (100 to about 800°C.).—Provided corrosion does not interfere, many requirements in this range are met by common structural metals, and of course many more by the heat-resistant alloys. In the lower part of the range certain earthy materials—fused silica, chemical earthenware, etc.—are available as corrosion resistants, and for mild corrosive conditions the common refractories are a final, often the most economical, resource.

Materials for corrosive use at ordinary temperature divide themselves roughly into three groups: the metals, of high strength but individually limited in application to specific corrosives; earthy materials (fused silica, earthenware, etc.), of moderate strength but broad chemical resistance; and a few carbonaceous materials (wood, asphaltum, etc.), neither strong nor widely resistant, but for certain uses economical.

Corrosion resistance from its nature cannot be measured in any one unit. Reference^b notes the resistances of various alloys to a few specific reagents, and occasional tests are reported elsewhere; these, with the recommendations of manufacturers and reports of successful chemical use, constitute a great part of the literature of the subject. This reference adopts a scale by which corrosion resistance is measured as "excellent," "good," "fair," or "poor," as the resistant material for a specific corrosive gives promise (1) of many years' life, (2) of a commercially long life, (3) of a practicable life where cost is not a consideration, or (4) of no useful life whatever. Many of these descriptions are quoted here, but the key words have been translated in terms of useful life as (1) "very long," (2) "long," (3) "moderate," or (4) "none."

THE METALS

Strength Properties at High Temperature.—As its temperature reaches melting point, a metal of course loses its strength altogether. At much lower levels it is

seriously weakened, and its elastic properties are profoundly modified. Under continuous load it tends to "creep," *i.e.*, to yield slowly by plastic deformation at unit stresses that are apparently within its elastic limit when rapidly tested at the same temperature in the laboratory. Fahrenheit^{1,c} even questions the existence of a true elastic limit at any temperature, though recognizing that under ordinary structural conditions the "creep" is too slow for practical consideration. He defines useful strength at high temperature as the range of stress in which deformation remains too slow to produce adverse results within the span of useful life, and sets upper limits to this range well within the corresponding values for yield point as determined by rapid testing. His strength-time-temperature relation, however, is very sparingly reflected in existing records of test; it is true that the test rate of loading is usually reported, but specific determinations of stress within which deformation is negligible are almost entirely wanting. Records are therefore confined to showing the strength-temperature relation, which the designer must translate into working stress by experience, or by applying high safety factors, or by trial and error.

Material.—Obviously, at temperatures of anneal the metals lose any increment of strength due to prior mechanical work, and French and Tucker^a note the waning benefit due to quenching observed in steels used at 450 to 600°C. Fahrenheit finds that time reduces the temperature necessary to weaken or destroy the temper of quenched material, and recommends a preparatory heating at temperatures higher than those to be met later in service. He regards alloys of the solid-solution type as the most useful in resisting heat, especially those of complete solubility in all proportions. Nevertheless, materials of diverse alloy type are offered for high-temperature service, as well as commercially pure metals.

Thermal expansion (Fahrenheit) is another factor in high-temperature design that under ordinary conditions plays a much less important part. If stress induced by expansion exceeds the limits of negligible deformation, some degree of permanent set is inevitable, and failure is possible. If heating is followed by cooling, contraction then induces a reverse stress and causes a reverse plastic deformation, doubling the original effect. The inference, of course, is that allowances for thermal expansion and contraction at high temperature should be proportioned to the magnitude of the temperature range and to variations in the rate of expansion as temperature is increased. Table 100 contains data on thermal expansivity of most of the materials treated in this subdivision.

High-temperature testing rarely reveals the full extent of "creep," and is liable to errors of observation absent in ordinary testing. There is no commercial—or even scientific—standard among methods of diffusing and measuring test temperatures, and the results of different operators are apt to diverge by more than the usual limits of work with the cold piece. Doubts have been raised as to some of the earlier results, but criticism itself is not yet precise, and much of the existing data must serve until experience proves it wrong. However, the warning sounded by French and Tucker^a for iron and steel may be extended to all metals—that strength values

¹ The lettered references in the text denote:

^a "Symposium on Effect of Temperature upon the Properties of Metals," with extensive bibliography, prepared for a joint meeting of the A.S.T.M. and A.S.M.E., May, 1924.

^b "Symposium on Corrosion-resistant, Heat-resistant and Electrical Resistance Alloys," with data supplied by the manufacturers, A.S.T.M., June, 1924.

^c "Some Principles Underlying the Successful Use of Metals at High Temperatures," Fahrenheit, *Proc. A.S.T.M.*, Vol. 24, (Part 2).

^d *Chemical and Metallurgical Engineering*, symposium on materials for equipment construction, Apr. 16, 1923

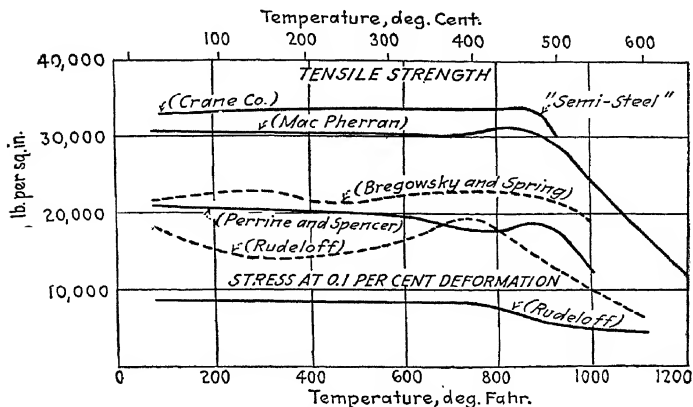


FIG. 31.—Cast and malleable iron.

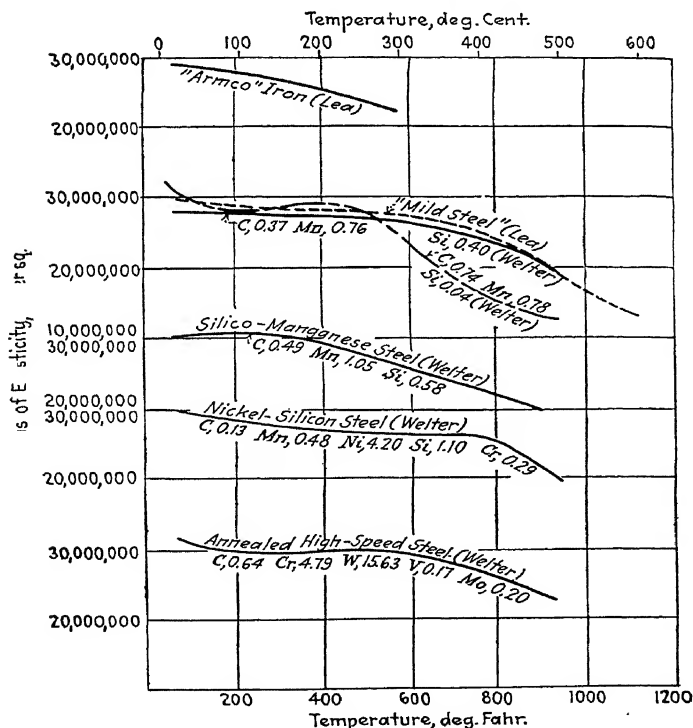


FIG. 32.—Properties of wrought iron and steel.

obtained in a single test represent only the conditions of loading employed in that test, and are not to be lightly generalized.

Cast and Malleable Iron (Fig. 31).—Cast iron, semisteel, and malleable iron retain strength practically unimpaired up to 425°C., but beyond 480° rapidly weaken. French and Tucker^a quote practical limits for cast iron of 260°C. and 25-lb. gauge pressure, or 40°C. and 75-lb. gauge; for semisteel and malleable iron, of 260°C. and 200-lb. gauge.

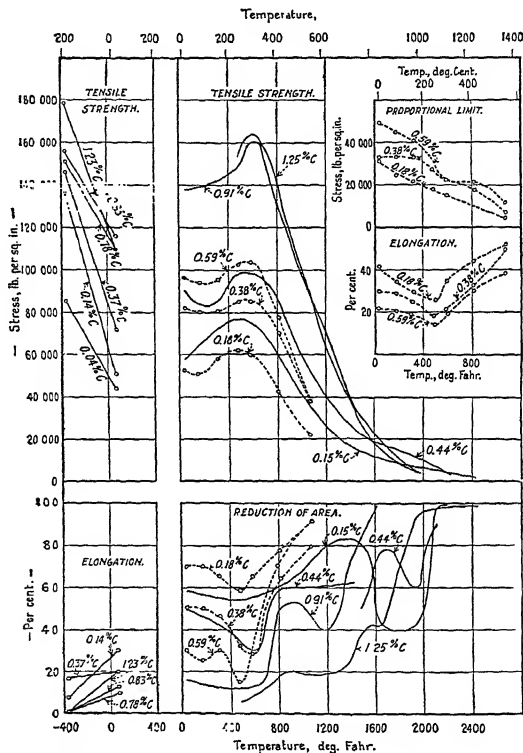


FIG. 33.—Effect of temperature on tensile properties of carbon steels.

Wrought Iron and Steel.—The conclusions of French and Tucker^a for worked steels (Figs. 32 to 34) are summarized below. In general, they hold for cast steels, though with variation in individual values (Fig. 35). Wrought iron acts much like low-carbon steel.

1. At 600°C. tensile strength, proportional limit, and elastic modulus are reduced, and ductility and "creeping" much increased. Carbon steel and many alloy steels are strongest and least ductile between 200 and 350°C.

2. In low- and medium-carbon steel, the proportional limit for internally stressed material (e.g., cold-worked or quenched) decreases comparatively little through a

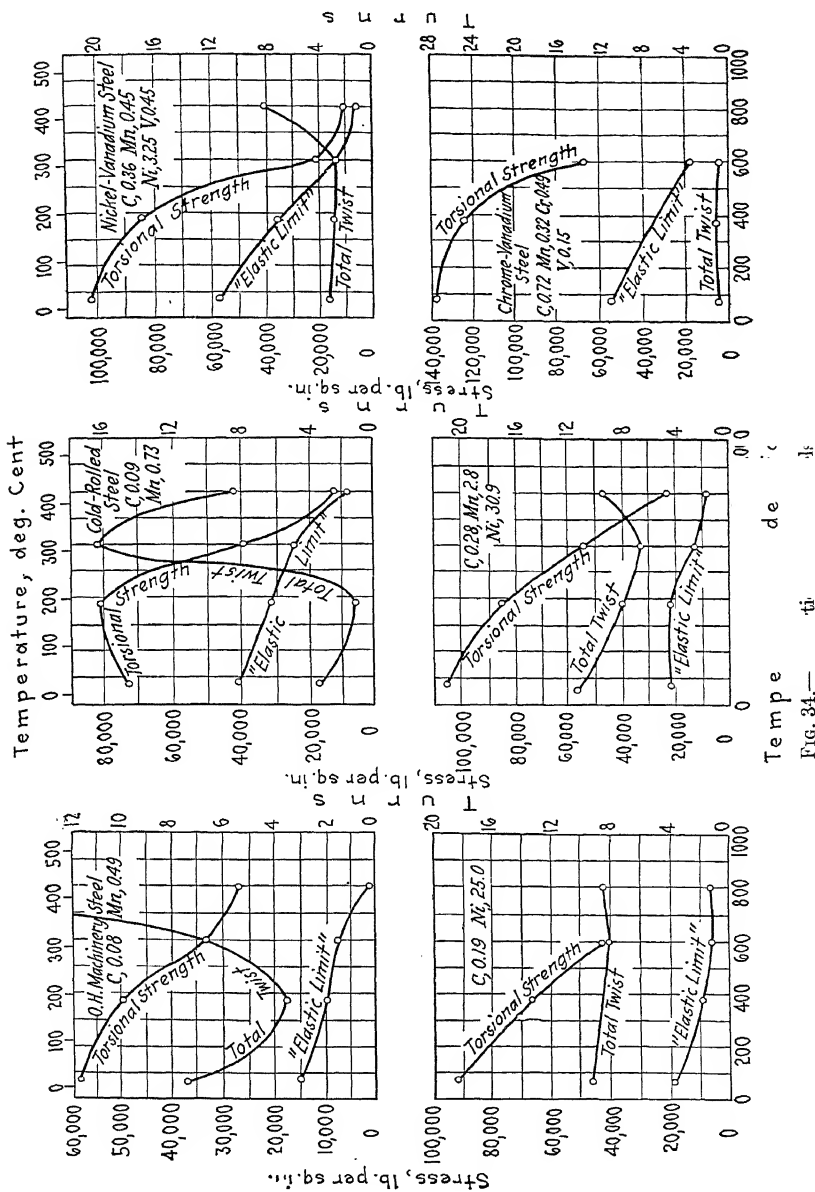


FIG. 34.—

considerable rise in temperature; for material in which internal stress has been relieved, it decreases more quickly.

3. Between 20 and 450°C., ultimate strength and proportional limit depend on composition and heat-treatment, and steels that are strongest at 20°C. usually remain so throughout this range; ductility is more apt to decrease in the high-carbon than in

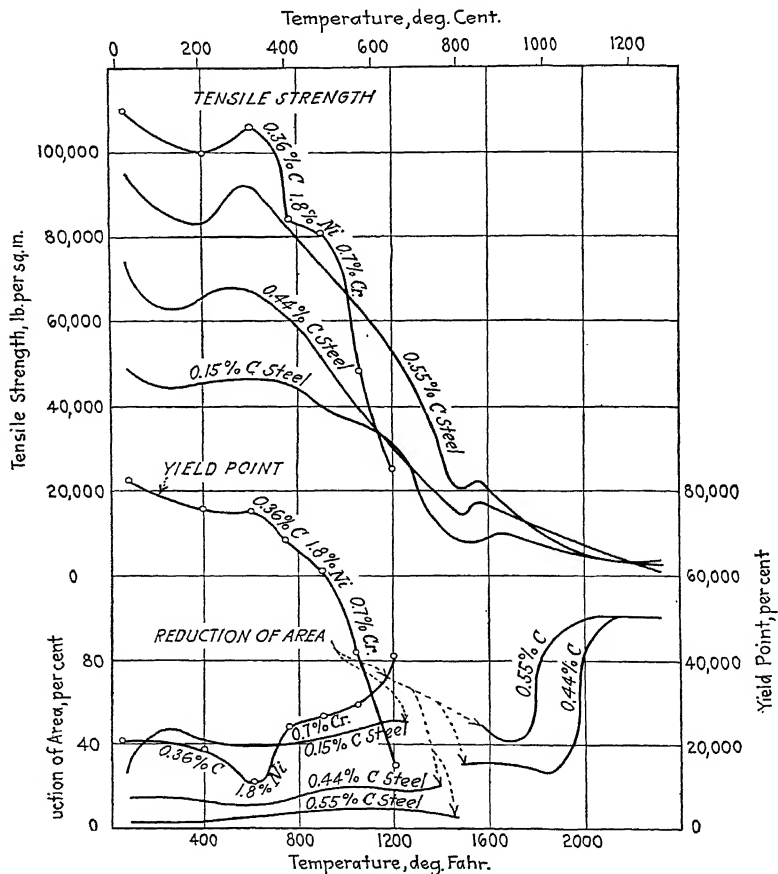


FIG. 35.—Properties of cast steel.

the low-carbon steels. Between 450 and 600°C. the benefit of quenching is less and less marked, as higher drawing temperatures are required to stabilize the strength of steel for high-temperature work, whence composition assumes more importance. Steels of high proportional limit should prove the stronger in this range, and alloying elements may, therefore, prove of value. Above 600°C. the decrease in ultimate strength and proportional limit is very marked, and commercial steels—except those

TABLE 84.—VARIOUS HIGH-CHROME STEELS

	Carpenter Steel Co.		Firth-Sterling Co.		Midvale Co.
	Stainless	Rustless	Stainless steel	Stainless iron	
Composition, 1 per cent:					Si, 0.35 per cent; Mn, 0.50 per cent
C.....	0.30	0.30	0.30	0.15	0.35
Cr.....	14.0	20.0	13.0	13.0	15.0
Tension (cold):					
Ultimate, 1,000 lb. per square inch.....	115-260	100	100-267	117-241
Yield point, 1,000 lb. per square inch.....	85-225	68	65-210	218 max.
Proportional limit, 1,000 lb. per square inch.....	50	221 max.
Elongation modulus, 1,000 lb. per square inch.....	30,000	30,000	30,000	66 min.
Endurance limit, 1,000 lb. per square inch.....	40
Elongation in 2 in., per cent.....	23-11	22	27-0	23-6
Reduction of area, per cent.....	59-32	50	59-1	61-11
Brinell.....	240-512	200	175-565	230-485
		?			17-1 35-3 230-493
Useful life in contact with reagent					
Reagent					
H ₂ SO ₄ :					
Diluted.....	None	None	None	Moderate
Concentrated.....	None	None	None	None
HNO ₃ : both.....	Very long	Very long	Very long	Very long
HCl:					
Diluted.....	None	None	None	Long
Concentrated.....	None	None	None	None
Acetic:					
Diluted.....	Very long	Very long	Long	Moderate
Concentrated.....	Very long	Very long	Very long	None
NaOH.....	Very long	Very long	Very long	Very long
NH ₄ OH.....	Very long	Very long	Very long	Long
Moist sulphurous atmosphere.....	None	Very long	None	None
Cl ₂ aqueous soluble.....	None	None	None	None
Gases with CO.....	Very long	Very long	None	Very long
Saturated steam.....	Resistant
Molten brass.....	Long

1 Balance, Fe and impurities.

2 Maximum temperature for continuous operation in air, 1000°C.

trace. On the time-honored basis of carbon content the latter are treated here with the steels, and the carbonless ferroalloys are grouped under their own caption.

High-nickel Steels.—Figure 36^a shows strength-temperature curves for certain 25 to 35 per cent nickel grades. Nickel alone does not give steel its greatest corrosion resistance, although its alliance with chromium produces in this respect perhaps the most effective alloy of all.

High-chrome Steels.—Table 84^b contains data on the corrosion resistance of stainless steels, of which the first four are recommended by their makers for chemical equipment.

Chrome-nickel Steels.—Of the Rezistal series (see page 550), Atha's 2600, and Rezistals 4, 7, and 8 have a total range of composition^b of 0.15 to 0.70 per cent carbon, 1.25 to 3.25 per cent silicon, 22 to 36 per cent nickel, and 5.5 to 27.5 per cent chromium. Table 85 has been compiled from reference^b and *National Bureau of Standards Circular* 100. The latter notes a 2-in. Rezistal piston rod used for two years in the liquefying cylinder of a liquid-air machine at a stress of about 3000 lb. per sq. in. Among other uses, the Rezistals are recommended for mine and chemical equipment, furnace bottoms, annealing and casehardening boxes, enameling and hardening trays, stirring rods, ice and liquid-air machinery, etc. *Number 17 Metal* (Cyclops Steel Co.^b) is a similar alloy containing 0.45 per cent carbon, 1 per cent silicon, 20 per cent nickel, and 7.5 per cent chromium, with listed strength properties comparable to those of Rezistal No. 8; its useful life in contact with dilute nitric and acetic acids is rated as very long, with dilute sodium and ammonium hydroxide very long, with dilute sulphuric acid and gases containing carbon monoxide long, with dilute hydrochloric acid and chlorine in aqueous solution moderate; it is usually attacked by sulphates and chlorides. *Calite* is another member of this group, used in the cast form (see Table 86). *Calite A* is resistant to molten lead, molten salts (except cyanide), carburizing compounds, and cool distillates, and is suitable for carburizing boxes, retorts, and hearth plates. *Calite B* is "used only for beams supporting loads at elevated temperatures."

TABLE 86.—CALITE

	A	
Composition, per cent:		
C.....	0.80	1.50
Ni.....	35	6
Cr.....	15	18
Fe and impurities.....	Bal.	Bal.
Ultimate tensile strength, pounds per square inch:		
20°C.....	60,000	60,000
815°C.....	30,000	30,000
Machining qualities.....	As for 0.80 C tool steel	None
Forging qualities.....	Forgeable with 0.2 per cent C maximum	None
Welding qualities.....	Good	None
Maximum temp. for continuous operation in air, degrees Centigrade.....	1150°	

Chrome-silicon steels are grouped in Table 87.

Silicon irons, to meet corrosive conditions, require a silicon content (according to Schenck^a) of at least 12 per cent. They are used as cast, and their corrosion resistance varies with minor variations of composition, pouring temperature, and cooling rate. They withstand nitric, acetic, and sulphuric acids in any concentration, hot or cold, but are attacked by hydrochloric and hydrofluoric acids, bromine, fused alkalis, concentrated hydrofluosilicic acid, and the chlorides of sulphur. For caustic

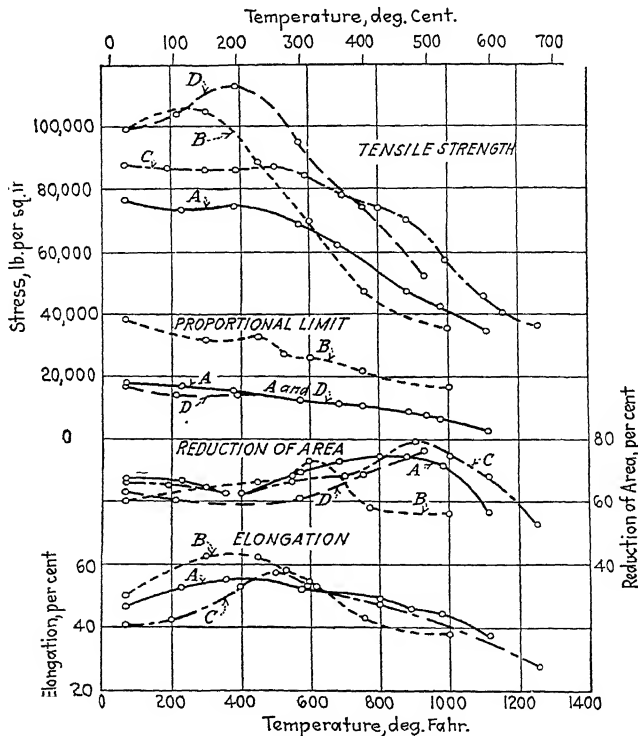


FIG. 36.—Properties of high-nickel steels.

solutions, sulphurous acid solutions, and certain sulphites they should be selected only after test.

Duriron and Corrosiron.—See Table 88 and the prior discussion of chemical resistance, which applies in detail to the properties of Duriron listed in reference,^b in which its resistance to arsenic salts, sulphates, chlorides, nitrates, phosphates, and organic acids is also noted; some test data are quoted in the table. Corrosiron is rated^b with very long life in contact with sulphuric, nitric, and acetic acids, sodium and ammonium hydroxides, and moist sulphurous atmosphere; long life in contact with hydrochloric acid of moderate dilution; and a moderate life with either very dilute or concentrated hydrochloric acid and chlorine in aqueous solution. Both alloys present difficulties

TABLE 87—CHROME-SILICON STEELS (Allegheny-Ludlum Steel Co.)

	Delhi tough iron	Delhi hard	Sichrome	Reagent	Useful life in corrosive service		
					Delhi tough	Delhi hard	Sichrome
Composition, ¹ per cent:							
C.....	0.07	1.20	0.40	H ₂ SO ₄ :			None
Si.....	1.25	1.50	3.50	Dilute.....	Moderate	None	Long
Cr.....	17.0	17.0	8.25	Concentrated.....	Long	Long	Very long
				HNO ₃ both.....	Very long	Very long	None
				HCl 16 per cent.....	None	None	None
Tension:				Concentrated.....	None	None	None
Ultimate ²	77-85	127-237	124-212	Acetic acid:	Very long	Very long	Very long
Yield point ³			86 min.	Dilute.....	Long	Long	Long
Proportional limit ³	45-50	80-187	74-177	Concentrated.....	Very long	Very long	Long
Elastic modulus ⁴	26,700	29,000	29,500	NaOH.....	Very long	Very long	Long
Elongation 2 in., per cent.....	50-25	17-0	25-2	NH ₄ OH.....	Very long	Very long	None
Reduction area, per cent.....	67-14	30-0	197 ²	Moist sulphurous atmosphere.....	Very long	Very long	None
Brinell index.....	183-170	207-600		Cl aqueous solution.....	Very long	Very long	None
Compression:				Gas with CO.....	Very long	Very long	Very long
Proportional limit ²	46-49	70-152	Pb at 930°.....	Very long
Maximum temperature of continuous operation in air.....	1000°C.	930°C.	950°C.	Pb at 1000°.....	Very long	Very long
				Air.....	930°C.....	Very long
				HNO ₃ boiling.....	Very long
				Acetic acid ⁵	Very long	Long	Very long

Strength tests at room temperature.
¹ Balance is Fe and impurities.
² Thousands of pounds per square inch.
³ Annealed.

Strength tests at room temperature.

Balance is Fe and impurities.

Thousands of pounds per square inch.

3 Annealed.

TABLE 88.—DURIRON AND CORROSIRON

	Corro- siron	Duriron			
			Loss in inches thickness per month at temper- ature (Centigrade) of		
				Room	93°
Composition, per cent:			H ₂ SO ₄ :		
C.....	0.60		0.5 per cent.....	0.0000057	0.000130
Si.....	13.5	14.25	20 per cent.....	0.0000039	0.000071
Mn.....	0.35	0.35	Concentrated.....	0.0000016	0.0000063
Fe.....	85.5	84.6	HNO ₃ :		
Compressive ultimate pounds			0.5 per cent.....	None	0.0000023
per inch.....	68,000	70,000	20 per cent.....	0.0000017	0.000033
			Concentrated.....	0.0000015	0.00016
Test at room temperature.			HCl:		
			0.5 per cent.....	0.00012	0.00140
			20 per cent.....	0.00667	0.028
			Concentrated.....	0.01822	0.1017
			Acetic acid:		
			0.5 per cent.....	None	None
			20 per cent.....	None	None
			Concentrated.....	0.0000015	0.0000023

in the casting of large plane surfaces; in the design of Duriron vessels open at the top, material near the rim is reinforced with heavy ribs, and, if necessary, steel inserts can be cast in place.

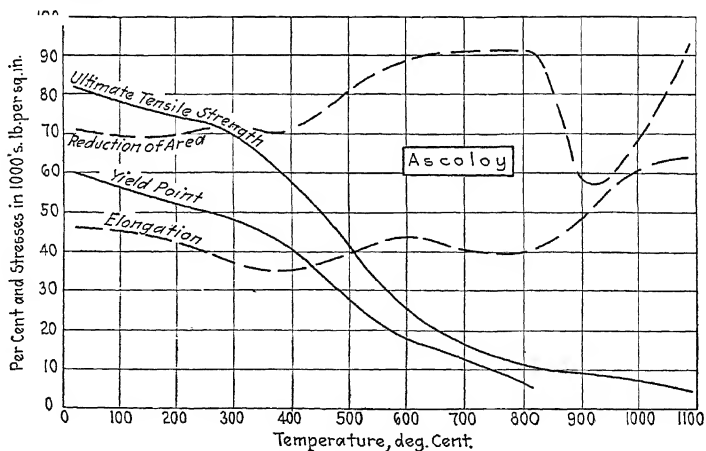


FIG. 36a.—Ascoloy constants.

Ferroalloys.—The Nichrome and chromel groups contain members in which iron is wanting and are treated among the nonferrous metals. Alloys that contain iron as an important constituent are treated here.

Iron-chrome Alloys (Table 89).—Ascoloy (14 per cent chromium)^b is marketed in rods, bars, sheet, tube, and wire. It is not recommended for temperatures above 800°C. (Fig. 36a). Chrome iron, Duraloy, Cimet^b (20 to 30 per cent chromium) are cast, forged, and made in bars, tubes, wire and sheet; they are unsuited to reducing atmospheres above 1100°C. and are less resistant to shock after long heating near 500°C.

Iron-chrome-nickel Alloys (Table 89).—Elalco Comet^b (30 per cent nickel, 5 per cent chromium) is used in dipping baskets and as an electrical-resistance material.

TABLE 89.—FERROALLOYS

	Ascoloy	Chrome iron, Duraloy, Cimet	Comet	Calido
Tension (cold):				
Ultimate, 1,000 lb. per square inch.....	73-180	70-120	85	100
Yield point, 1,000 lb. per square inch.....	40- 70	47	56
Proportional limit, 1,000 lb. per square inch	40-160			
Endurance limit, 1,000 lb. per square inch.	35	25		
Elastic modulus, 1,000 lb. per square inch.	27,200			
Elongation in 2 in., per cent.....	37- 15	30- 0	37*	28*
Reduction of area, per cent.....	79- 51	50- 0	69	59
Brinell index.....	150-365	170-400		
Maximum temperature of continuous operation in air, degrees Centigrade.....	800°	1100°	700°	1000°
Reagent	Useful life in contact with reagent			
H ₂ SO ₄ :				
Dilute.....	None	None	Long	Very long
Concentrated.....	Long	None	Moderate	Long
HNO ₃ :				
Very dilute.....	None	Very long	Moderate	Long
Dilute.....	Long	Very long	Moderate	Long
Concentrated.....	Very long	Very long	None	Long
HCl:				
Dilute.....	None	None	Moderate	Long
Concentrated.....	None	None	Moderate	Moderate
Acetic acid:				
Dilute.....		Long	Long	Very long
Concentrated.....	Very long	Long		Long
NaOH:				
Dilute.....	Very long	Long	Very long	Very long
Concentrated.....		Long	Very long	Very long
NH ₄ OH.....		Long	Very long	Very long
Cl in aqueous solution.....	Long	None	Moderate	Moderate
Gases containing CO.....		Very long		
Phosphoric, citric, fruit acids..		Very long		
Molten silver.....		Very long		
Molten brass, salt niter baths.		Long		
Moist sulphurous atmosphere..		Long		
Manufacturer.	Allegheny- Ludlum Steel Co.	Cutler Steel, Driver- Harris Cos., licensees	Electrical Alloy Co.	

It is produced in wire, rod, ribbon, and strip. Calido (60 per cent nickel, 16 per cent chromium) is made in the same forms; it is embrittled by heating in contact with silicates.

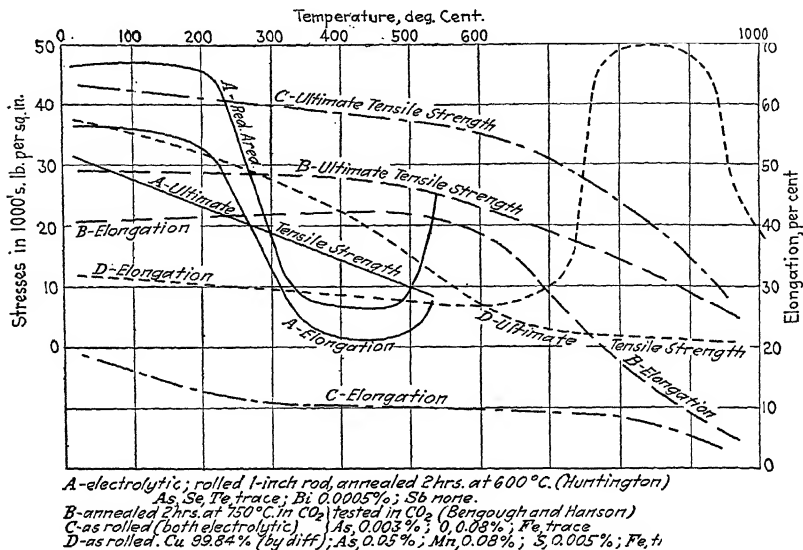


FIG. 37.—Constants for copper.

TABLE 90.—CORROSION RESISTANCE OF VARIOUS BRASSES

Useful life in contact with	Naval brass	Admiralty metal	Commercial bronze rod
H_2SO_4 :			
Very dilute.....	Long	Long	Very long
Dilute.....	Moderate		Long
HNO_3 :			
Dilute.....	None	None	
Concentrated.....		None	
HCl :			
Very dilute.....	Moderate	Moderate	Long
Dilute.....	Moderate	None	
Concentrated.....		None	
$NaOH$, dilute.....	Long	Long	Long
NH_4OH , dilute.....		Long
Moist sulphurous atmosphere.	Long	Moderate	Very long

Copper under atmospheric conditions retains useful strength through a limited temperature range; its strength properties are apparently sensitive to composition,

condition (cold rolled or annealed), and chemical surroundings, as the tests in carbon dioxide indicate (Fig. 37). It finds use in applications of organic chemistry, such as the manufacture of many esters, ethyl acetate, and alcohol, and in organic chemistry, such as acetone, methyl acetate, dimethyl

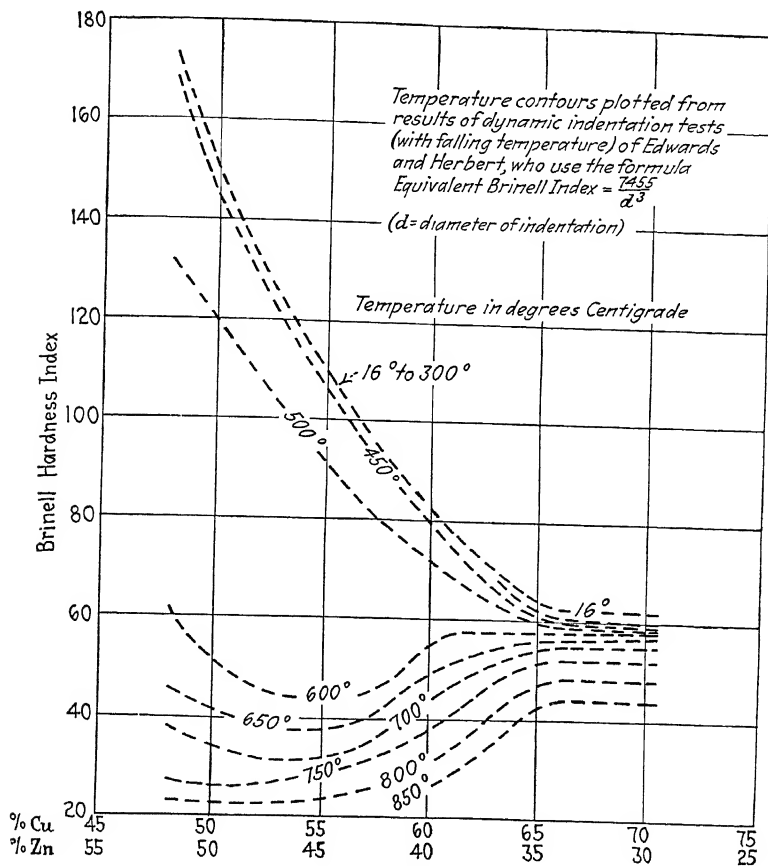


FIG. 38.—Constants for brasses.

subject to attack by maleic, fumaric, lactic, tartaric, and malic acids and by ammonia and nitric acid (Steffens^a).

Brasses and Bronzes.—Both of these alloy groups are more serviceable as steam metals than in the higher temperature range, and more characteristically resistant to ordinary rusting than to severe chemical corrosion. Nevertheless, they are cheap and easily obtained and, for certain purposes, are capable substitutes for the more refractory special alloys.

Brasses.—Figure 38 shows hardness-temperature relations for copper-zinc alloys (Edwards and Herbert),¹ and Fig. 39 contains strength-temperature curves for various common brasses, Fig. 40 for manganese bronze. Table 90 contains data on the corrosion resistance of Naval brass, Admiralty metal, and "commercial bronze rod"

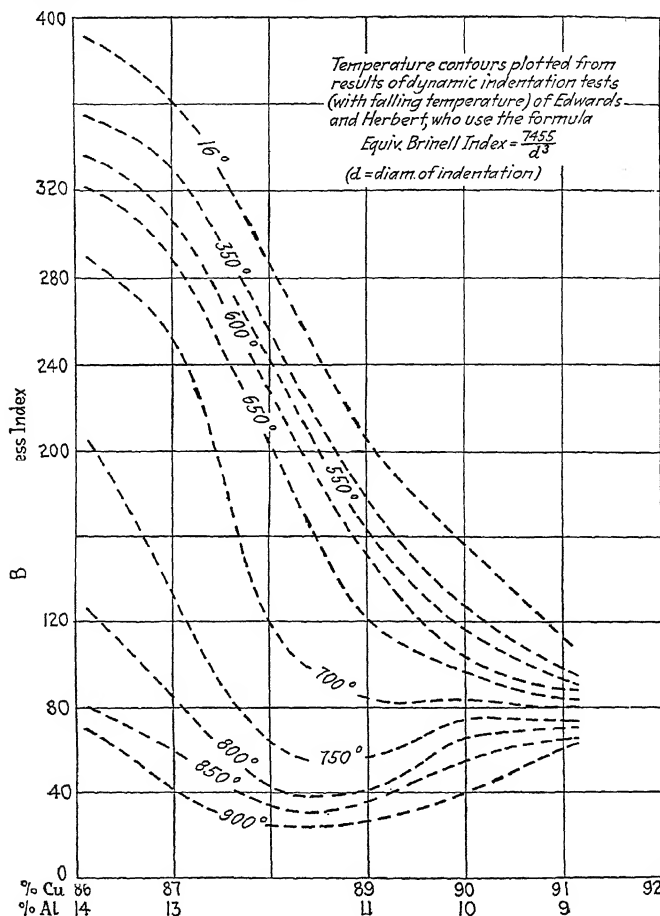


FIG. 38a.—Constants for aluminum bronze.

(Chase Metal Works). After immersion for 5 months, rolled Tobin bronze is reported by the American Brass Co. to have lost 0.04 per cent of its weight in saturated salt solution, 2.95 per cent in full strength, and 1.49 per cent in a 50 per cent solution of U.S. standard ammonia, 0.25 per cent in one-third strength sulphuric acid, and 0.06 per cent in caustic potash. Whittaker² recommends manganese bronze for 85

¹ Jour. Inst. Metals, Vol. 25, p. 175.

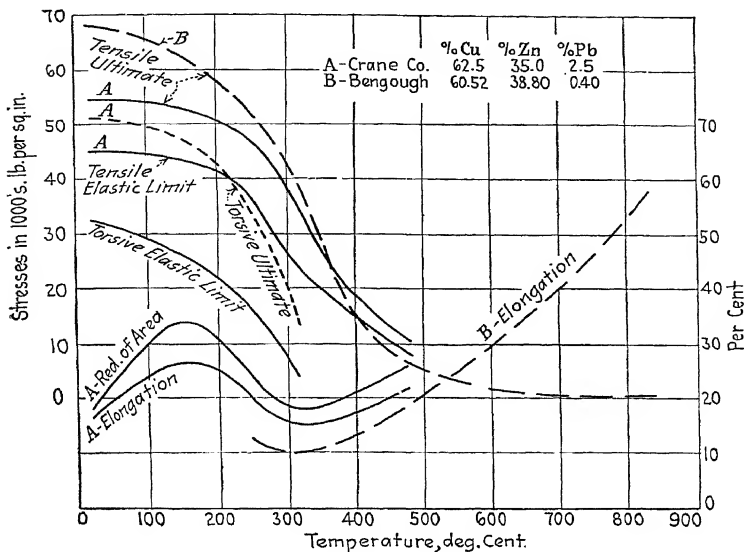


FIG. 39.—Properties of brass.

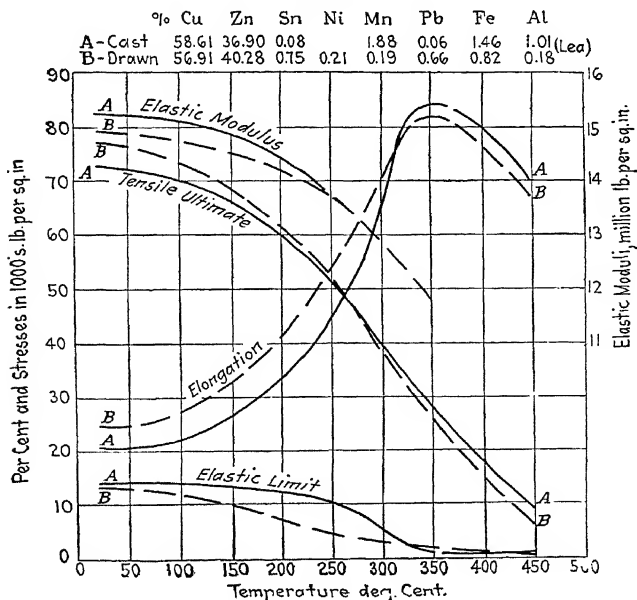


FIG. 40.—Properties of Mn bronze.

per cent formic acid and notes its resistance to warm dilute (1 to 2 per cent) sulphuric acid; but he condemns the use of brass and bronze in handling ammonium nitrate, because of their effect on the reagent.

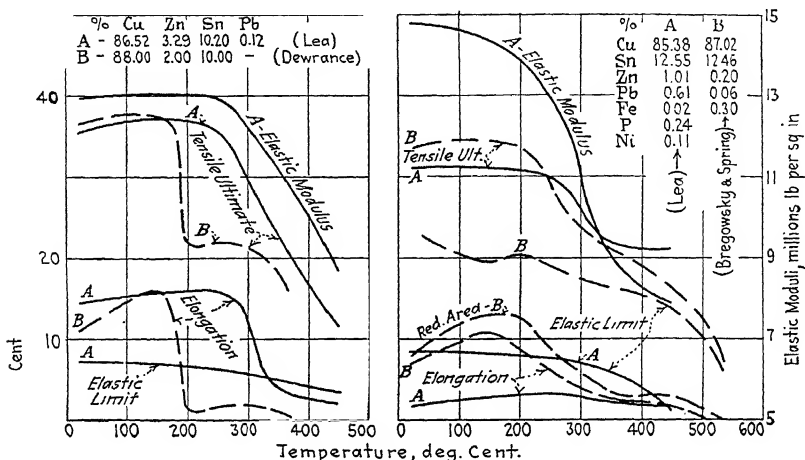


FIG. 41.—Gun metal and phosphor bronze.

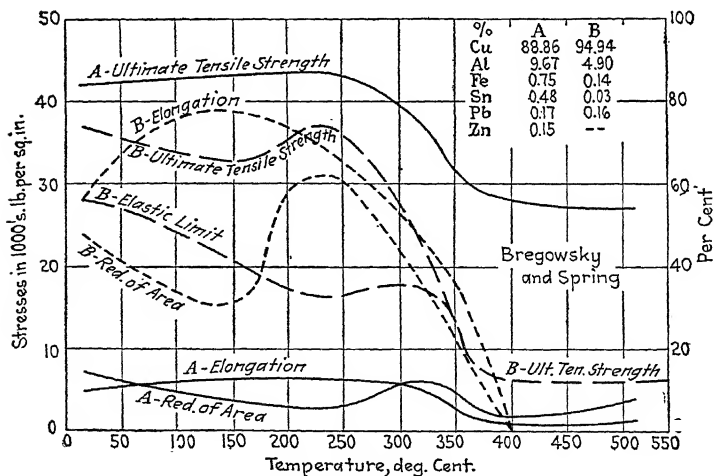


FIG. 42.—Aluminum bronze.

Bronzes.—Figure 41 shows strength-temperature curves for gun metal and phosphor-bronze, and Fig. 42 for aluminum bronze. Two titanium-aluminum bronzes (Nos. 1 and 5, Frontier Bronze Corp.²) are reported as resistant to sulphuric and hydrochloric acids; one contains 90 per cent copper and 10 per cent aluminum, the

other is a variant, with 1 per cent of the copper replaced by iron. Ampco^b is listed as containing 82 per cent copper, 10 per cent aluminum, and 8 per cent iron and is "recommended for many chlorides, sulphates, and salts of organic acids, also many of these acids, not for oxidizing acids or mixtures of H_3PO_4 "; its useful life under exposure to sulphuric acid and moist sulphurous atmosphere is reported as very long, to hydrochloric and acetic acids as long, to ammonium hydroxide as moderate, to nitric acid as slight.

Copper-silicon Alloys.—Everdur No. 50 (Dupont Engineering Co.^b) contains 94.5 per cent copper, 4.5 per cent silicon, and 1 per cent manganese. It is made in castings, forgings, sheet, rod, wire, etc., and is recommended for equipment connected with the handling of sulphuric and hydrochloric acids and their salts. Table 91 shows its listed properties, to which may be added its freedom from permanent expansion or growth on heating between room temperature and 535°C.

TABLE 91.—EVERDUR No. 50

Tension	Cold range	20°C.	260°C.	400°C.	535°C.
Ultimate, pounds per square inch.....	56,000-143,000	73,000	60,000	36,000	14,500
Yield point, pounds per square inch.....	48,000				
Elongation in 2 in., per cent.....	56-0	44	33	26	18
Reduction of area, per cent.....	53-0	51	67	69	71
Corrosive penetration, inches per month, 25°C. unagitated		Useful life, exposed to			
Sulphuric acid, 20 per cent.....	0.001	Nitric acid.....	None		
Sulphuric acid, 70 per cent.....	0.004	Acetic acid.....	Very long		
Hydrochloric acid, 20 per cent.....	0.023	NaOH.....	Very long		
Hydrochloric acid concentrate.....	0.224	NH ₄ OH.....	Moderate		
Cl (aqueous soluble) 12.5 grams per liter.....	0.16	Moist sulphurous atmosphere	Very long		

Lead.—The well-known use of chemical lead (see page 508) as a resistant to sulphuric acid requires comparative freedom from dirt and oxide rolled into the sheet,

TABLE 92.—CORROSION RESISTANCE OF NICKEL

Thickness of scale after two weeks in moving air. Tested at temperature (degrees Centigrade) of					Loss in weight (milligrams per square inch)		
					Immersed, days	7	28
600° 0.001 in.	700° 0.001 in.	800° 0.001 in.	900° 0.004 in.	1000° 0.008 in.	Solution	Renewed	Not renewed
Corrosive penetration, inches per month							
H ₂ SO ₄ , 5 per cent, 38-49°C.....					0.00141		
HNO ₃ , 5 per cent, 38-49°C.....					0.00635		
Acetic acid, 5 per cent, 100°C.....					0.00197-0.00214		
Loss in weight, milligrams per square inch after 2.5 hr. in boiling acetic acid, 0.01 to 0.04							
					H ₂ SO ₄ (N/5).....	16.1	25.8
					HNO ₃ (N/5).....	271.0	135.5
					HCl (N/5).....	16.1	29.0
					NaOH (N/5).....	0	0
					NH ₄ OH (N/5).....	0	0
					NaCl (N/5).....	0	0
					CaCl ₂ (N/5).....	5.2	3.2
					MgCl ₂ (N/5).....	3.2	6.4
					Na ₂ CO ₃ (N/5).....	0	0

from bismuth as an impurity in the metal, and from segregated areas of impurities.¹

Antimonial lead, containing 4 to 10 per cent antimony, is credited with better service than the chemical grade below 105°C., though inferior at higher temperatures. Its strength is greater, especially as to elastic limit, and it is lighter (Hoyt Metal Co.). It is supplied in sheet and pipe of the sizes customary for chemical lead (see Table 51);

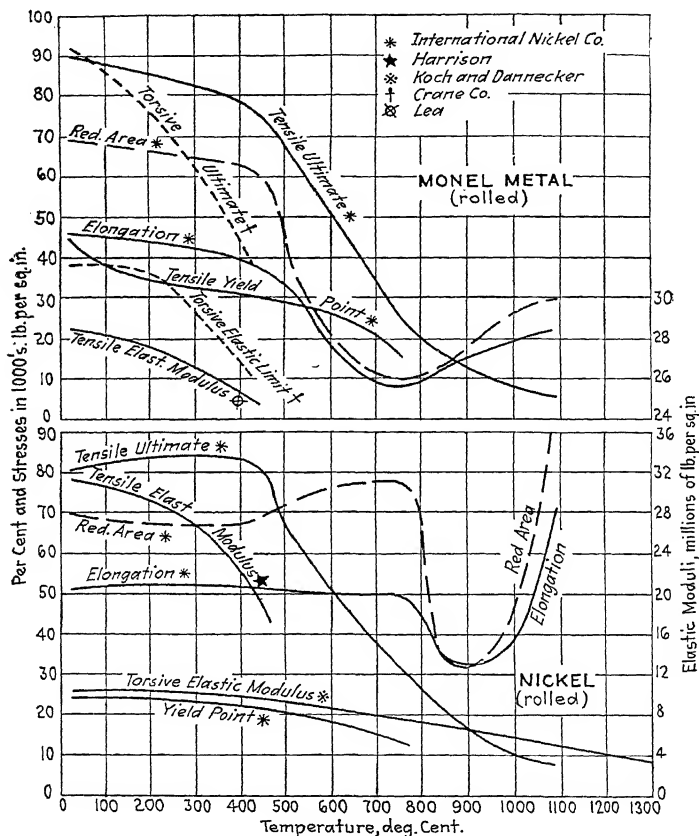


FIG. 43.—Properties of nickel and Monel metal.

the 6 per cent antimony grade finds frequent use. *Tellurium lead* has the property of toughening when strained. This gives it a great utility in such equipment as agitator tanks when vibration stresses are encountered. Its corrosion resistance exceeds that of ordinary lead, as well as greater tensile strength.²

Nickel is available in many fabricated forms, distinguished for their smooth finish. Strength-temperature curves for the commercial metal are shown in Fig. 44, and

¹ FRENCH, *Chem. & Met. Eng.*, Vol. 27, p. 219.
² *Dutch Boy Quart.*, Vol. 13, p. 15, 1935.

Table 92 contains data on its corrosion resistance (*National Bureau of Standards Circular 100* and reference^b); it is also recommended for use with caustic alkalis, carbonates, cyanides, chlorides, etc., but not with molten metals. At temperatures above 800°C., it is preferable to Monel metal, and is credited with a limit of continuous operation in air of 1300°C.

Nickel Alloys.—A glance at these pages will suggest the very general use of nickel as an element of heat- and corrosion-resistant alloys. Those treated here include all alloys in which the metal plays an important part, except the ferrous.

Monel Metal.—See Fig. 43 for strength-temperature relations and Table 93 for corrosion resistance (*National Bureau of Standards Circular 100* and reference^b). This alloy is not recommended for use in air at temperatures above 800°C. It is "not appreciably attacked by dry gaseous or liquid ammonia, dilute or concentrated ammonium hydroxide solutions (intermediate concentrations attack it slightly), (used caustic alkalis or carbonates or their aqueous solutions, fatty and many other organic acids, sea water, solutions of many neutral salts (including sulphates and chlorides), gasoline and other mineral oils, phenol and cresols, photographic chemicals (except silver-bearing solutions) . . . it resists well the action of sulphuric, dilute phosphoric,

TABLE 93.—CORROSION RESISTANCE OF MONEL METAL
(Solutions not aerated)

Corrosive penetration, inches per month		Loss in weight				
		Per	Milligram per square inch			
H ₂ SO ₄ , 1 per cent, boiling.....	0.0034					
HNO ₃ , 0.7 per cent, boiling.....	0.02					
HCl:						
0.3 per cent, boiling.....	0.00314					
1 per cent, ^a room temperature..	0.00046					
20 per cent, ^a 80°C.....	0.25					
Acetic acid:						
1 per cent, boiling.....	0.0021	Boiling acetic acid:	Hour	0.07		
99 per cent, boiling.....	0.04-0.11	10 per cent.....	Hour	0.08		
NaOH:		26 per cent.....	Hour	0.11		
2.5 per cent, boiling.....	0.000044	56 per cent.....	Hour	0.12		
20 per cent, boiling.....	0.000004	90 per cent.....	Hour	0.12		
		Cold glacial acetic acid.....	Day	0.076		
Thickness of scale after 2 weeks in moving air. Tested at temperature (Centigrade) of						
		600°	700°	800°		
		900°	1000°			
		0.001	0.005	0.008		
		0.220	0.250			

^a Not agitated.

hydrocyanic, hydrofluoric, acetic and citric acids, fused cyanides, ferrous sulphate, and dry chlorine; it is not resistant to the action of hydrochloric or nitric acid, molten lead or zinc, sulphurous acid, ferric chloride, or chromic acid.¹ It is used for tank and crate parts in pickling steel, and specimens have resisted corrosion in this service for years. Other examples include pump liners, rods, and valves in contact with acid and alkaline solutions, various parts required in the dye, chemical, and oil industries, and filter cloth for corrosive solutions.

¹ National Bureau of Standards.

Modified Monel Metal (M-M-M).—For strength-temperature relations up to 482°F., see Table 58.

Other Nickel-copper Alloys.—As an example of the constantan type (see page 513), a strength-temperature relation for Advance (Driver-Harris Co.^b) is shown in Fig. 44; this is essentially a 45 nickel-55 copper alloy used in rheostats, resistance spools, and thermocouples, and supplied in wire, ribbon, and sheet. The same figure shows a curve for Everbrite No. 92 (Curtis Bay Copper and Iron Works^b), an alloy of 35 per cent nickel and 58 per cent copper supplied in bars and forgings for oil distillation, steam- and gas-engine valves, and chemical equipment. Everbrite No. 90 (35 per cent nickel, 60 per cent copper) is made for chemical-plant valves and pump parts. Adnic (see page 514), aside from its use in thermostats, is made in tube, sheet, and wire

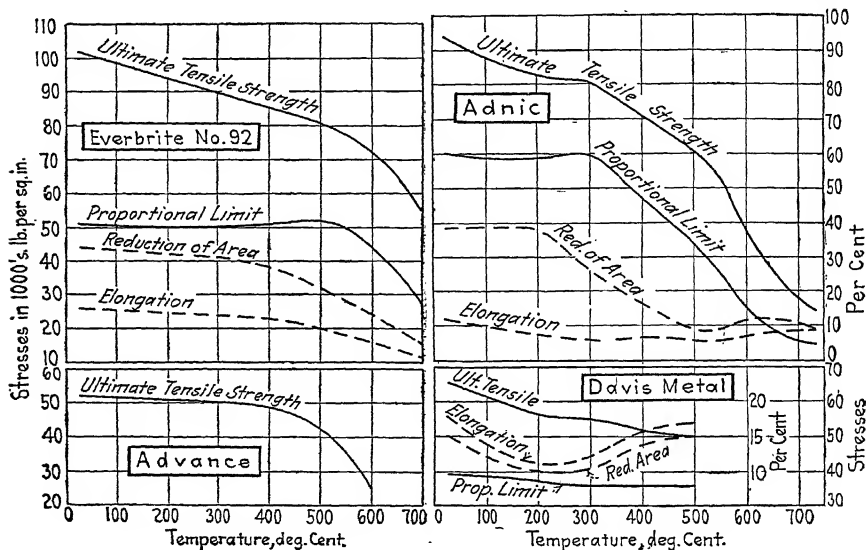


FIG. 44.—Nickel-copper alloys.

for various purposes, *e.g.*, that of oil-still condensers; its strength-temperature curve^b appears in Fig. 44, and corrosion losses^b are recorded for the annealed metal of 0.18 to 0.52 mg. per sq. in. per day in 2 per cent sulphuric acid, 0.40 to 0.32 in the 5 per cent acid, both at room temperature unagitated, the first result in each case applying to metal of softer temper. Davis metal used in castings (Chapman Valve Mfg. Co.^b) contains 29 per cent nickel, 67 per cent copper, and 2 per cent iron with minor constituents; its strength-temperature curve is shown in Fig. 44 and corrosion tests in Table 94; it is listed as not attacked by formic, oxalic, citric, tannic, boric, or common fatty acids at 66°C., or by fuel oil, dry chlorine, or many organic volatile solvents at normal temperature. Aterite includes several alloy compositions (Aterite Co. Inc.,^b) and these, with strength properties, are shown in Table 95. Its corrosion resistance is recorded as indicating very long service with sulphuric acid, moist sulphurous atmosphere, and sodium hydroxide, negligible service with hydrochloric and nitric acids and ammonium hydroxide (both wrought and cast metal included in each case).

TABLE 94.—C

	Solution, temperature degrees Centi- grade	Agitation, inches per minute	Penetration, inches per month
H ₂ SO ₄ , 10 per cent.....	60-66	2.5	0.00035
H ₂ SO ₄ , 30 per cent.....	60-66	2.5	0.00027
HCl, 10 per cent.....	60-66	2.5	0.0091
NaOH, 10 per cent.....	60-66	2.5	0.00046
NH ₄ OH, 10 per cent.....	60-66	2.5	0.00081
Cl (aqueous solution).....	21	0	0.0044
HF, 48 per cent.....	21	0	0.00017
Chromic acid, 60 per cent..	71	2.5	0.00027
H ₃ PO ₄ , 85 per cent.....	21	0	0
Picric acid.....	21	...	0
Picric acid.....	77	2.5	0.0048
Al ₂ (SO ₄) ₃ , 13 per cent.....	66	2.5	0
HgCl ₂ , 10 per cent.....	60	2.5	0.0196
FeCl ₃ , 10 per cent.....	60	2.5	0.0300
NaCl, 10 per cent.....	66	2.5	0.000024
Ammonium phosphate.....	60	2.5	0
Ca(OH) ₂ , 10 per cent.....	60	2.5	0.00019
H ₂ SO ₃	21	2.5	0.000033

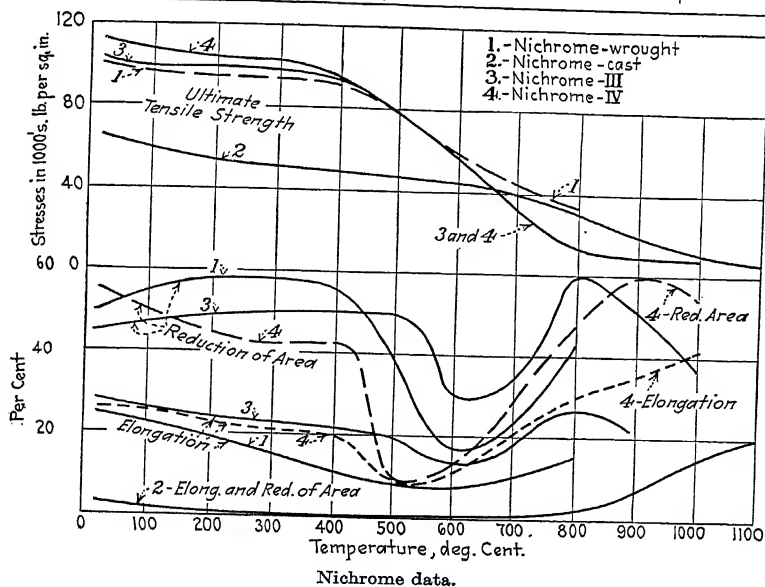


TABLE 95.—ATERITE

	Rods, bars, wire, sheet	Castings
Composition, per cent:		
Ni.....	10	35-44
Cu.....	65	55-36
Fe.....	2	5-20
Zn.....	23	5- 0
Tension:		
Ultimate, pounds per square inch.....	53,000-163,000	55,000-110,000
Yield point, pounds per square inch.....	40,000-110,000	35,000-110,000
Endurance limit, pounds per square inch....	24,700	17,600
Elongation in 2 in., per cent.....	59- 1	45- 0.6
Reduction of area, per cent.....	62-17	36-10
Torsional shear:		
At maximum load, pounds per inch.....	65,000	55,000
At proportional limit, pounds per inch.....	30,000	18,000

TABLE 96.—NICHROME

		Nichrome		Nichrome III, wire, ribbon, sheet, forgings	Nichrome IV, forg- ings, wire, rod, strip
		Wire, strip, rod, sheet, forgings	Castings ¹		
Composition, per cent:					
Ni.....		60	66	85	80
Cr.....		12	16	15	20
Fe.....		Bal.	Bal.		
Cold range	Tension				
	Ultimate, pounds per square inch.....	100,000	65,000	103,000	112,000
	Yield point, pounds per square inch.....	60,000	55,000	57,000	63,000
	Elongation 2 in., per cent.....	25	3	28	26
	Reduction of area, per cent.....	50	2	45	55
Comp.	Ultimate, pounds per square inch.....	211,000		
	Proportional limit, pounds per square inch...	35,000		

Useful life in service with reagent:

HNO ₃	None
HCl:	
Dilute.....	Moderate
Concentrated.....	None
Acetic acid..	None
NH ₄ OH.....	Very long

¹ Nichrome castings only:

H ₂ SO ₄ :	Loss, grams per square inch per hour at 20°C.
Specific gravity 1.830.....	0.00007
Specific gravity 1.747.....	0.00019
Specific gravity 1.408.....	0.000
Specific gravity 1.142.....	0.000
Specific gravity 1.036.....	0.00006
H ₂ SO ₄	0.00042

Chrome-nickel alloys include groups of ranging composition in which iron and copper may appear as secondary constituents.

Nichrome Group (Driver-Harris Co.^b).—Table 96 shows the characteristics of several members of this group: the composition first listed is recommended for pickling machine parts, valves, annealing pots, heating units, etc.; the second (forged as well as cast) for dipping baskets, carburizing boxes, and retorts; Nichrome III and IV are much used in high-temperature heating devices and are credited with better chemical resistance than Nichrome. The results of corrosion tests on the casting alloy are shown in Table 96; the two Nichrome compositions are not recommended for hot sulphur-bearing gases or sodium compounds at high temperature (especially those containing Na_2CO_3), but are rated as serviceable with molten tin and silver and with lactic acid.

Chromel Group (Hoskins Mfg. Co.^b).—These alloys are supplied in rod, wire, strip, flats, and castings. Chromel C and No. 502 are recommended for corrosion resistance, the latter (castings) in furnace parts and carburizing boxes; all four are recommended for heat resistance, and A, B, and C are used in electrical heating devices. Chromel C is rated as very resistant to molten lead and white metals, and fairly resistant to cyanide and other molten salts; it is suitable for stirring molten brass and aluminum, but not as material for their containers.

TABLE 97.—CHROMEL

	A	B	C	502
Composition, per cent:				
Ni.....	80	85	Bal.	25
Cr.....	20	15	11	20
Fe.....			25	55
Tension, cold range:				
Ultimate, pounds per inch.....	93,000–119,000	88,000–97,000	30,000
Elongation in 1.5 in., per cent.....	52	47	
Elongation in 6 in., per cent.....	25	16	
Maximum temperature (Centigrade) of continuous use in air.....	1150°	1100°	1000°	
Corrosive penetration, inches per month				
Acetic acid:				
Medium dilution.....	0.0003			
Concentrated.....	0.0030			
H ₂ SO ₄ :				
Medium dilution.....	0.0004	–0.0005		
Concentrated.....	0.0009	–0.0010		
HNO ₃ :				
Medium dilution.....	0.01	–0.07		
Concentrated.....	0.0008	–0.0030		
HCl:				
Medium dilution.....	0.0005	–0.0030		
Concentrated.....	0.05	–0.06		
NH ₄ OH, medium dilution.....	0.000001–0.00001			

Ilium (Standard Calorimeter Co.^b) is a complex alloy of 63 per cent nickel, 21 per cent chromium, 8 per cent copper, 4 per cent molybdenum, 2 per cent tungsten, 1.5 per cent manganese, and 0.5 per cent silicon, used in cast form for calorimeter bombs and pump parts. Its ultimate tensile (cold) is quoted at 85,000 lb. per sq. in., elongation

in 2 in. at 5 per cent and reduction of area 2 per cent; for its corrosion resistance, see Table 98.

TABLE 98.—CORROSION RESISTANCE OF ILLIUM
Loss in milligrams per square inch per month

Reagent.....	H ₂ SO ₄			HNO ₃			HCl			Acetic acid	H ₃ PO ₄		
Strength.....	10 per cent	25 per cent	95 per cent	10 per cent	25 per cent	70 per cent	5 per cent	10 per cent	25 per cent	99 per cent	10 per cent	25 per cent	85 per cent
Temperature, degrees Centigrade													
20-30	1.87	1.03	0.19	0.19	0.52	1.55	32.9	224	0	0.26	0.13	0.04
70-75	23.9	49.7	32.9	11.6	22.6	29.7	851	1,071	3.87	21.0	7.1	4.1

Useful life very long with NaOH, NH₄OH, moist sulphurous atmosphere, CO.

No useful life with chlorine in aqueous solution.

Aluminum (see Chap. I of the volume entitled "Recovery of the Metals").—The commercial metal is attacked by hot dilute sulphuric acid, slowly by the concentrated; its resistance to cold nitric acid is sufficient for practical use, but it is very easily attacked by hydrochloric and hydrofluoric acids, chlorine, bromine, iodine, and alkali hydroxides. Acetic and butyric acids act on it slowly when cold, more rapidly as temperature and dilution increase. Lactic and oleic acids have very little effect.

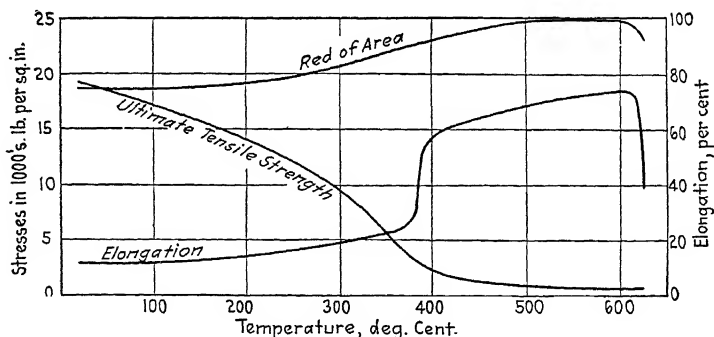


FIG. 45.—Properties of aluminum.

Aluminum is used (Blough²) for crystallizing pans for ammonium sulphate, tartaric acid, and citric acid; for gasoline tanks, vinegar containers, turpentine and wood-distillation stills, wax-cooling pans, rubber-curing pans, and various parts employed in the vegetable oil industry. Its strength-temperature relations are of less importance than those of stronger materials; a curve obtained by Bengough¹ is shown in Fig. 45.

Aluminum Alloys.—Figure 46 shows the strength-temperature relations recorded in the eleventh Report to the British Alloys Research Committee, which will serve as

¹ Jour. Inst. Metals, Vol. 7, p. 123.

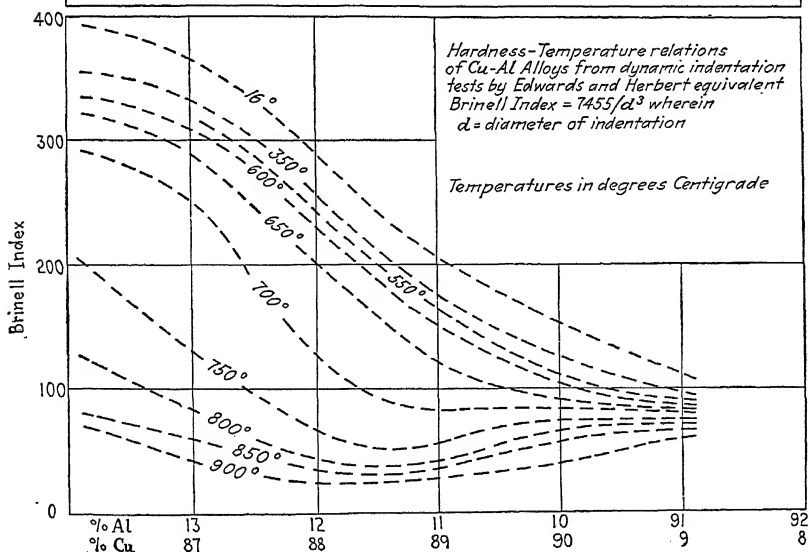
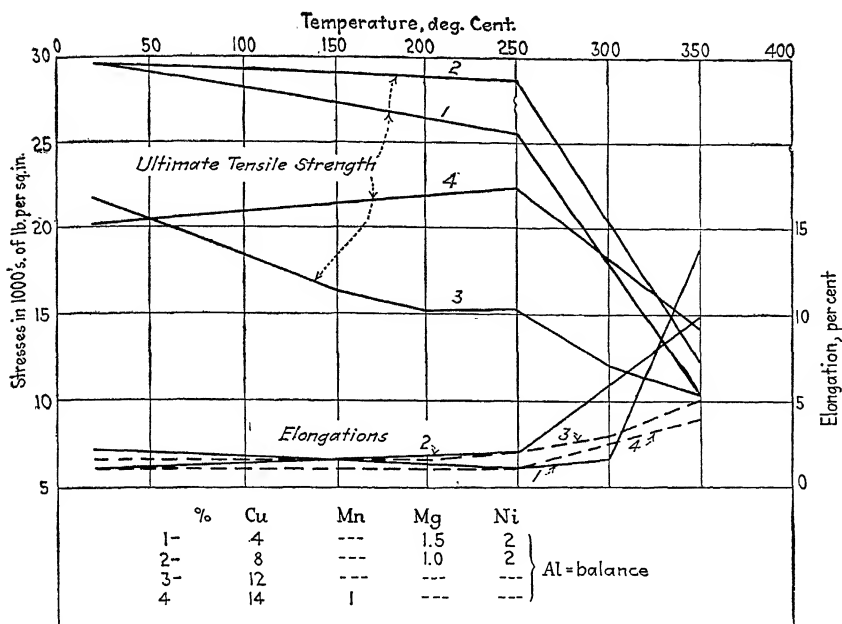


FIG. 46.—Aluminum alloys.

a reference for the subject. The hardness-temperature relations as found by Edwards and Herbert¹ are shown in Fig. 46.

EARTHY MATERIALS

General.—Enameled coatings receive a certain "reinforcement" from the metals to which they are applied, but fused silica, chemical stoneware, and glass are typical earthy products, weak in tension and more or less brittle in the useful range of temperature. They are not materials for severe and complex stress conditions, but they have enough strength to make their marked corrosion resistance available in units of respectable capacity, and they retain this strength through a considerable range of temperature.

Enameled equipment² supplies a link between the metals and the earthy materials, and is available in units as large as tank equipment for railroad stock. The coated metal is steel or cast iron, the former in larger pieces for mild corrosive conditions, the latter more often in small units with more resistant enamel.

Heating Units.—Obviously, under corrosive conditions the enameled coating must remain unbroken, and as its thermal expansivity cannot be adjusted to that of the metal throughout a wide range of temperature, it is apt to crack under severe local heating. Therefore the direct firing of an enameled unit requires care and uniform distribution of heat. Heating by means of jackets or coils within the unit is safe. Among the circulating media are steam and hot oil, although steam pressure must be kept within limits, owing to the contrast in ductility between enamel and metal. A pressure of about one-half the working limit for the uncoated metal is allowable. Jones³ places the pressure limits for a cast-iron enameled unit at 75 lb. gauge in jackets and 100 lb. within the unit, and for a steel-enameled unit at 30 lb. in the 500-gal. and 10 lb. in the 3000-gal. size, with jacket temperatures by the use of steam up to 160°C. (if superheated, 300°), and by the use of oil up to 225°.

Enamels are classified by Poste as suitable for (1) neutral or very slightly corrosive materials—alcoholic extracts, aniline hydrochloride, ammonium nitrate during evaporation and fusion, bleach liquor, chlorine gas, dry hydrogen-chloride gas, hydrogen peroxide, phenol, salicylic acid, acetic anhydride, dry sulphur dioxide, tannic acid, bromine; (2) weak acids—citric and tartaric acids, acetic acid—acid brines, moist chlorine, mixed acids relatively low in water content; (3) strong acids—nitric, hot hydrochloric—and dilute mixed acids; (4) alkalis, of which the hydroxides and sulphides of sodium and potassium are difficult to handle, although ammonia, caustic soda solutions of less than 5 per cent, and (in certain concentrations) carbonates of sodium and potassium offer possibilities. Table 99 shows typical formulas for the raw materials of enamel, as quoted by Jones; the "ground coat" is the first to be applied, the "cover" the finishing coat. The ground coat for cast iron consists chiefly of quartz and borax.

Uses.—In concentration or evaporation to dryness, enameled equipment is suited to small quantities and special corrosive conditions, rather than rapid operation with large volumes. Mixing operations are well within its scope, and enameled mixing or agitating devices are also available. In general, as corrosive conditions become more severe, the design of enameled equipment must be simplified and its size reduced.

Chemical stoneware⁴ consists of a skeleton of refractory particles in a binder of vitrifying clay. It is made in remarkably complex forms and units—coils, fans (for handling corrosive gases), pumps, pipes, branches and fittings, cylinders, etc., indi-

¹ *Jour. Inst. Metals*, Vol. 25, p. 175.

² *POSTE, Chem. & Met. Eng.*, Vol. 27, p. 1016.

³ *Chem. & Met. Eng.*, Vol. 25, p. 927.

⁴ *KINGSBURY, Chem. & Met. Eng.*, Vol. 57, p. 1226.

TABLE 99.—BATCH FORMULAS FOR ENAMEL

Raw materials	Steel		Cast-iron cover
	Ground coat	Cover	
Soda.....	74	...	71
Borax.....	356	154	133
Saltpeter.....	...	65	
Chilean saltpeter.....	24
Lead oxide.....	144
Zinc oxide.....	44
Tin oxide.....	73
Calcspars.....	...	65	24
Barium carbonate.....	11
Magnesium carbonate.....	...	10	
Feldspar.....	363	386	372
Fluor spar.....	53	13	107
Cryolite.....	...	117	
Quartz.....	144	190	
Manganese dioxide.....	6½		
Cobalt oxide.....	2¼		
Mill additions:			
Clay.....	6 %	7 %	
Borax.....	2.5 %		
Tin oxide.....	...	12 %	
Magnesium oxide.....	...	0.25 %	

eating some capacity for withstanding rough usage. Its chemical resistance varies to some extent with composition; in general, it is effective with all acids except hydrofluoric and hot strong phosphoric (*i.e.*, the corrosives of silica), but is not immune to hot concentrated caustic alkalies.

Composition.—A typical analysis appears in Table 100, which also contains a general statement of physical properties. To overcome porosity, the ware is usually

TABLE 100.—CHEMICAL STONEWARE

Composition, per cent		Physical properties	
SiO ₂ ..	73.23	Ultimate tensile strength, pounds per square inch..	1,000–2,200
Al ₂ O ₃	22.27	Ultimate compressive strength, pounds per square inch.....	25,000
CaO.....	0.58	Modulus of elasticity, pounds per square inch.....	6,000,000–9,000,000
MgO.....	tr	Specific heat.....	0.2
K ₂ O.....	2.02	Thermal conductivity, B.t.u.....	0.55–0.6
Na ₂ O.....	1.42	Specific gravity.....	2.17
Ignition loss.....	0.06	Thermal expansivity per degree Centigrade.....	0.00003–0.000005

(These values are averages, subject to variation in the individual piece.)

given a salt glaze, typically a sodium aluminum silicate of roughly 66 per cent silica, 20½ per cent alumina, and 12½ per cent sodium oxide.

Glass is impervious, easily fabricated in small forms, and, of course, has the special property of transparency. It has been used in the form of small rings as a tower packing,¹ in pipe lines, etc.; the glass tubes of the Hart condenser, 6 ft. long and 3 in. in diameter, show the sizes possible in plant service. Its well-known liability to hydrofluoric acid attack and imperfect resistance to strong alkalis are among its

TABLE 101.—THERMAL EXPANSIVITY AND CONDUCTIVITY OF HEAT- AND CORROSION-RESISTING MATERIALS
(All temperatures are centigrade)

Material	Thermal expansivity					Thermal conductivity		
	Range of temperature	Coefficient					Temperature	Calories per square centimeter per second per centimeter of thickness per degree Centigrade temperature difference
Admiralty metal.....	0-100°	0.0000202						
Asteroite:								
Wrought.....	0-100°	0.0000165					Room	0.077
Cast.....	0-100°	0.0000148					Room	0.045
Cast.....	140°	0.0000185					140°	0.055
Bronze rod, commercial.....	0-100°	0.0000185						
Calido.....	0-100°	0.000013						
Calite A & B.....	Room-815°	0.000006						
Chrome iron, Duraloy, Cimet.....	0-800°	0.0000119						
Chromel A, B, C, No. 502.....	20-1000°	0.000016						
Cimet:								
Wrought.....							Room	0.082
Cast.....	0-800°	0.0000153						
Comet.....	0-100°	0.000016						
Davis metal.....	0-100°	0.0000144					Room	0.045
Duriron.....	0-100°	0.0000292					Room	0.125
Everdur.....	25-100°	0.000017						
Fused silica.....	200°	0.00000518						
	900°	0.00000538						
	1100°	0.00000583						
Invar.....	0-40°	0.000001						
Glass:								
Hard.....		0.0000077						
Soft.....		0.0000085						
Pyrex.....		0.0000032						
Illium.....	1100°	0.0000135						
Monel metal.....	25-100°	0.000014					Room	0.06
	25-300°	0.000015						
	25-600°	0.000016						
Naval brass.....	0-100°	0.0000214						
	25-100°	0.000013						
Nickel.....	25-300°	0.0000145						
	25-600°	0.0000155						
Nichrome, wrought and cast.....	0-100°	0.0000122					Room	0.033
Nichrome III and IV.....	0-100°	0.0000132					Room	0.033
Rezilal 4, 7, and 8.....	20-300°	0.0000161						
	300-600°	0.0000180						
	600-900°	0.0000202						
Rustless steel (Carpenter).....	23-250°	0.0000099						
	250-500°	0.0000105						
	500-700°	0.0000104						
Stainless steel (Firth-St.).....	20-200°	0.0000109					Room	0.0445
Titanium-aluminum bronze.....	0-100°	0.0000171						
Nickel:								
Temperature.....	-200°	0°	200°	400°	600°	800°	1000°	1200°
Conductivity coefficient.....	0.125	0.140	0.135	0.116	0.088	0.068	0.065	0.057

¹ Chem. & Met. Eng.

chemical limitations; toward most other corrosives it is very resistant. Its shock brittleness and tendency to fail under localized heating are laboratory commonplaces that hold true for structural use; Greene² notes the weakening effect of scratches and cracks.

*Pyrex*¹ is a borosilicate glass of low expansivity, softening at about 800°C. Up to boiling temperatures it is resistant to mineral acids (except hydrofluoric and phosphoric) and is used in acid-distillation sets, nitric acid condenser tubes, pipe lines, small tanks, condensers for organic liquids, etc. It is supplied in flanged and socket piping, dishes and flasks, as well as in special shapes. Specific gravity, 2.25; specific heat, 0.2; thermal conductivity, c.g.s. unit, 0.0027; thermal expansivity, 0.0000032 per degree centigrade.

Fused silica² owes its low thermal expansivity, and therefore its ability to withstand violent temperature changes, to vitrification. It is devitrified by prolonged heating between 1150 and 1400°C. In the form of combustion tubes, it may be heated to 1100°C. under pressures as high as 4 atmospheres. It resists all mineral acids except hydrofluoric and (above 400°C.) phosphoric, and in its usual commercial form has very low thermal conductivity, owing largely to the inclusion within the silica of minute gas bubbles acquired during manufacture.

Plastics.—Beginning with Dr. L. H. Baekeland's discovery of the peculiar properties of a mixture of phenol and formaldehyde, a whole new race of materials of construction has grown up in the so-called "plastics." The metallurgical engineer is less concerned with them than is the chemical, but it is well to have them in mind if a tank must be proofed against corrosion, or if fans or pumps, where there are not too great stresses, are to be resistant to corrosive agents. In general, they fall into the classes of phenolic, cresylic, urea, styrol, or vinyl resinoids, or of some cellulose product. Their number grows by dozens each month, and in a work of this sort it is hopeless to try even to catalogue them. The engineer is merely advised to remember that they exist and if he appears to have a problem to which they apply, to write to the manufacturers.

Structural Carbon.—Carbon blocks, laid in carbon cement, furnish a structure that resists solvent action, resists oxidation, has a low coefficient of thermal expansion, and is fairly resistant to mechanical abrasion. The joints must be baked in at 250 to 300°C. to get a completely integral structure.

Weight per cubic foot, 100 lb. (porosity, 25 per cent).

Tensile strength, 600 lb. per sq. in.

Crushing strength, 4000 lb. per sq. in.

Transverse strength, 1000 to 1500 lb. per sq. in.

Thermal conductivity, 0.00786 cal. per sq. cm. per °C.

Thermal expansion, 0.00000072 per °C.

Special electric resistance, 0.0042 to 0.0017 per in. cube.

Maximum safe temperature under oxidizing conditions, 350°C.

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CHAPTER XVI

PLANT LAYOUT

BY PERCY E. BARBOUR¹

Preliminary Considerations.—Before a plant can actually be laid out four decisions have to be made: (1) that a plant should be built; (2) its type; (3) its size; (4) its site.

Probably in the majority of mines the best engineering advice that can be given on the subject of the first consideration is "don't build." The world is full of idle metallurgical plants fallen or falling to decay because there was no original justification for their having been built. More mining properties have come to disaster through building an unneeded or a premature plant than by wrong design or inefficiency of the plant itself.

The building of a metallurgical plant is justified only when the value of ore blocked out in the property is sufficiently large to repay the investment in the mine, the cost of the mill, the cost of mining and treating the ore and marketing the product, the cost of administration, and a profit. Some of these points are often overlooked and, when they are not, are almost always underestimated.

The value of the ore blocked out must be carefully computed not on present quotations of the metals produced, but on the expected selling price over a period of years covering the life of the property. The starting point for such a calculation is, obviously, the average over past years, but the effect, if any, on the market for the product in question from known and also from partially developed sources, must not be overlooked, nor should changing economic and physical conditions.

Building a mill for stock-market purposes or to have it ready when the development work shall have blocked out sufficient ore is in one case an economic, and in the latter case an engineering, crime.

Assuming that a proper decision has been reached to build a plant, the type of plant is naturally the next point for decision. A discussion of this point is not within the scope of a textbook. Certain processes are today applicable to certain kinds and types of ores and certain processes are not; but metallurgy has changed overnight several times during the present generation and the end is not yet. The actual construction of a plant may be left to amateurs or unskilled hands with less risk of failure than the choice of type, which should be left to the judgment of an engineer experienced in the treatment of similar problems. In no other way can a reasonable chance of avoiding an irreparable mistake be assured.

The size of the plant to be constructed is a matter of calculation, or at least always should be, except for those very small plants which are inexpensive, rule of thumb, temporary, and simple from a metallurgical standpoint. Only on this principle can the proper size of a mill be determined, other than in the exceptions noted.

In his "Wealth and Income of the American People," Walter Renton Ingalls says: "The fundamental principle of mine valuation is the present worth of an annual dividend accruing during the period of years corresponding to the life of the mine."

¹ Deceased in government service, 1943.

This principle has been most succinctly amplified by Edwin S. Berry,¹ who shows the relation of the ore body to the size of the mill required and the relation of the size of the mill to earning capacity.

"Of two properties having the same grade of ore and the same operating conditions, the one having the greater ore reserves is not always the one having the greater value, as plant capacity enters into the calculations. In some cases, the ultimate practicable size of the plant may be absolutely limited by physical conditions, labor situation, market conditions, or other causes.

"Ore reserves can be made valuable only by building, or arranging for, a reduction plant to handle them, and the measure of their value is very largely dependent on the capacity of the plant. After a certain point has been reached, added ore reserves may mean little or practically nothing so far as present value is concerned, unless plant capacity can be increased.

"The life of a property depends on its reserves of profitable ore and on the size of its treatment plant, and the larger its treatment plant, and consequently the shorter its life, the greater will be its present value relative to the estimated total operating profits, assuming, of course, that the plant has been paid for.

"Generally speaking, the present value of a mine increases with its ore reserves. However, with a given plant capacity designed for the treatment of a certain tonnage of ore blocked out, further increase in reserves does not increase the present value nearly so much as might be supposed. As a general rule, reserves which give a life of about 30 years are found to be the limit beyond which the present value is not greatly added to unless plant capacity is increased. With the same ore reserves, however, the difference between the value of a property having a plant capacity that would give it, say, a 60-year life, and the value of the same property if its plant would give a 30-year life, is very marked. The figures in Table 1 will emphasize and make clearer the above remarks.

TABLE 1.—EFFECT OF PLANT CAPACITY AND ORE RESERVES ON PRESENT VALUE

Ore reserves, tons	Plant capacity, tons per day	Life, years	Profit per ton	Annual profit	Total operating profit over life	Present value	Per cent of total operating profit represented by present value
26,000,000	5,000	15	\$2	\$ 3,500,000	\$ 52,000,000	\$ 27,700,000	52.3
52,000,000	5,000	30	2	3,500,000	104,000,000	36,700,000	35.3
104,000,000	5,000	60	2	3,500,000	208,000,000	42,200,000	20.3
104,000,000	10,000	30	2	7,000,000	208,000,000	73,400,000	35.3
208,000,000	10,000	60	2	7,000,000	416,000,000	84,400,000	20.3
208,000,000	20,000	30	2	14,000,000	416,000,000	146,800,000	35.3
208,000,000	40,000	15	2	28,000,000	416,000,000	221,600,000	52.3

NOTE.—All present value figures in this paper are based on 8 per cent dividend and on 5 per cent interest on the amortization fund compounded annually.

"It will be seen from these figures that the shortest life shown, 15 years, gives the greatest present value relative to the total operating profits, and that even with such a short life the present value represents only a little over half of the total operating profits. It is of interest to note that, to consider the total estimated operating profits alone, without figuring the present value of such profits, would give a very erroneous idea of the value of the property.

¹ BERRY, "Present Value in Its Relation to Ore Reserves Plant Capacity, and Grade of Ore," *Mining Met.*, July, 1922.

"It will be seen also that to double ore reserves from, say, 104,000,000 to 208,000,000 tons without adding to the plant, and thus doubling the life, would add only about \$11,000,000, or 15 per cent, to the present value, equivalent to only about 11 cents per ton of added ore. On the other hand, to double the size of the plant for 104,000,000 tons of reserve, from 5000 tons per day to 10,000 tons per day, and thus reduce the life from 60 to 30 years, would add about 75 per cent, or about \$31,000,000 to the present value, from which addition the cost of added plant would have to be deducted, and would increase the annual profits from \$3,500,000 to \$7,000,000.

"The figures of 208,000,000 tons of ore reserves are included to show what enormous differences can be made in present value, and in annual profits, by increasing the size of the plant in mines having very large reserves, and what comparatively very small differences are made by great increases in ore reserves, if plant capacity is not increased to take care of such added tonnage.

"No matter how much the ore reserves are increased, the annual profits will remain the same unless the size of plant is increased also, but with no increase of ore reserves annual profits will increase more than proportionately with increase of plant capacity—more than proportionately because costs will usually be lowered when greater tonnage is handled."

Continuous processes are usually more economical than cyclic. Large units are usually more economical than small, though this does not always follow, for, as a mechanical roaster of the McDougall type goes above 20 to 25 ft. in diameter, the cost of maintenance increases greatly.

In figuring on the size of units, the effect of having all one's eggs in one basket must be considered, and the effect on the process of shutdowns for repairs and cleaning of the various units must be carefully considered. Often such consideration will dictate a subdivision of units that would otherwise not be made.

Vertical extension as well as horizontal should sometimes be considered; *i.e.*, where it is possible that extension of operations may sometime require an overhead crane, the building posts may be made strong enough to support it, even though all the headroom to the building may not be put in at that time.

Building should be done, but usually is not, when the price of metals is low and a condition of financial stress exists. To build in a time of prosperity means an interest charge that endures so long as the plant exists, and also often means the completion of a plant just in time to have a panic put a quietus on its operations.

Economic Factors Governing the Location of Plants.—Iron ores are freighted from South America to Pennsylvania for the manufacture of iron and steel; copper concentrates are freighted from Peru to Tacoma, Wash., for smelting; Bolivian tin concentrates are freighted from Bolivia to Corpus Christi, Tex., for smelting; some mills are built near the mines that produce the ores, and in some cases the ores are hauled long distances to mills to be beneficiated. This does not mean, however, that the designer of a plant has a free choice as to its site. Its location is entirely a matter of calculation of the various economic factors involved.

The choice of site for a plant depends on seven factors: raw material, water, power (electricity or fuel), supplies, labor, products, disposal of waste products (including fumes).

Transportation might be included to make an eighth item, but it is the determining factor in each of the other items except supplies. Supplies is the least important of the seven, and the transportation of supplies is negligible compared with the transportation of the raw materials and fuel. Any transportation that brings these two elements in can be depended upon to take the product out.

The first consideration, of course, is the raw material that is to be treated and from which the final product is to be made. An ample supply is the *sine qua non* which

has already been discussed. Whether the plant shall be located near the source of the material or the material brought to a more or less distant plant site depends entirely on the other factors. The location of blast furnaces and steel plants will be discussed under a separate head. These paragraphs are devoted to the consideration of nonferrous metallurgical plants. In these, where the product is bullion or base metal, pig zinc, lead, or copper, the market for the product has less effect in determining the site of the plant than in the case of steel, but latterly it does have an effect. For example, the copper refineries of the Anaconda and Phelps Dodge in Montana and Texas, respectively, supply copper for the Pacific coast and export without a long haul to the New Jersey refineries and back.

The most important consideration is water, because it can be provided only by nature, and therefore its sources of supply are limited and definitive. In some arid countries the only possible location for a treatment plant will be near the mine, where the mine water will be the only source of this supply. This is probably most applicable to the small treatment plants, both because the quantity of mine water is comparatively limited, and because with larger operations the cost of transporting water can be absorbed. An example of this situation is the New Cornelia operation at Ajo, Ariz., in a particularly arid desert country, but the operations are of such magnitude that water is obtained by sinking shafts 600 ft. deep and pumping the water 6 miles through a 20-in. pipe, with a very expensive equipment, to the mill site which is located near the mines themselves. It was cheaper to sink these well shafts and install expensive pumps and pump this water to the ore than to haul the ore to the Gila River, 50 miles away, where an ample water supply can be had the year around.

The quantity of water required for a plant is often underestimated, and in every calculation a most liberal factor of safety should be introduced into the quantity figure.

The use of electricity for power depends on its availability and on the prices per horsepower that can be contracted for. This will depend entirely upon the location and the policy of the power company, whether it is liberal or otherwise. If the power must be generated at the plant, fuel must be transported to it and, except in small operations in this country or new operations in virgin countries where wood may be considered, coal or fuel oil must be provided. Whether the plant shall be located near the source of the supply of the raw material and the fuel be brought to it or whether the plant shall be located near the main line which transports the fuel is a matter of dollars and cents to be determined by calculation. Consideration in such a case must be given to the fact that a very large percentage of the raw material will be rejected at the plant either as tailings or as slag, but, nevertheless, transportation from the mine to the mill must be paid on these materials.

Labor is an important factor, but is the most flexible of any, because with the modern methods of housing and recreating labor ample supplies of it can be drawn to the most isolated localities under ordinary circumstances. Present wartime conditions are, of course, entirely abnormal, and it is to be hoped will never occur again.

There is another phase of the labor factor in the item of mechanization. Cheap labor is always expensive labor. On the other hand, the mechanization of plants where labor is cheap and abundant may be carried to an extreme. This applies, of course, particularly to smaller operations, because in larger operations the higher cost of the mechanization is made necessary by the fact that, while the low grade of labor might be cheaper, the small army required would defeat the very purpose for which the plant was constructed.

Operating supplies other than fuel, as stated above, are a comparatively unimportant factor in determining the site because of their relatively small volume and the

fact that the transportation problem is determined by very much greater quantities of raw materials and fuel.

When branch lines have to be constructed from a main line to the plant, their cost must be amortized within the life of the plant or property, because, when the ore is exhausted, the branch line will have only scrap value represented by the rails, fittings, and the rolling stock, which will not be great, even though the operating efficiency and condition, if there were further ore to be hauled, may be very high.

Another important factor is the disposal of waste. The mineral laws of this country provide for the location and patenting of mill sites, in which is included the necessary ground for tailing disposal. Except in the very largest plants the question of this waste disposal does not affect the location of the plant except within small limits, but in plants like the Utah Copper, where 90,000 tons a day or more must be disposed of, this factor assumes a very great importance.

Plants that have a waste disposal containing cyanide or other harmful or poisonous ingredients must consider the people and livestock on the downstream side of the mill, or suits may arise involving damage and riparian rights. Plants that emit obnoxious fumes must guard against smoke suits. This can be done either by installing expensive mechanical or chemical means of rendering the gases innocuous, or by constructing the plant of such type, and in such localities, that the fumes will be so diluted and disseminated before they reach any land capable of producing crops that the danger of smoke suits may be eliminated.

The industrial engineers and efficiency engineers think they have discovered a new science because they have introduced into industry a straight-line flow sheet, known in the practice of metallurgy for a generation. Insofar as the contour of the site will permit, the plant should have a straight-line flow sheet. In the western states the country is so rugged that a plant may be built on a sloping site so that straight-line flow sheet will have eliminated from it most of the reelevating of material or solutions which costs so much money that it is incomprehensible that the foot-pounds of work expended in lifting these materials is often totally ignored. There are plants falling to decay in this country today which are located on a flat site, so that the material had to be lifted from the cars into one treatment machine after another and finally elevated from the last one to the tailing dump, when there are hillside locations that might well have accommodated the plant within rifle shot of such a monument to poor engineering.

Technical literature is poor in reliable cost data of metallurgical plants, and such figures as have been published almost always leave out the important figure of unit rates, which makes it impossible to translate the cost figures of one year and locality into the cost figures of another period of higher prices and in another locality of more difficult conditions.

When the size and the type of mill and the locality of the site have been determined, the detailed work begins and consists, first, in a preliminary layout of the surface dimensions and profile of the plant. The profile must have been assumed in a general way in the tentative site selection. Accurate contour and profile maps of the proposed site are made, on which are laid out the preliminary designs of the plant, and these maps are manipulated until the best adjustment is made and the site finally determined. Some of the factors involved in this determination are the amount of excavation necessary, the facilities for bringing in the raw materials and taking out the product, the landing of building material and supplies, the water supply, the disposal of the waste, and the reclaiming of the water, if that be necessary.

The final site selected, accurate final surveys are made, and contour and profile maps. The preliminary features of the plan of the mill may now be eliminated and

the main features definitely decided upon, whereupon the location may be staked on the ground and actual excavation may begin.

Except in minor operations it is not often that the first or second preliminary plan can be adopted. Sometimes a dozen have to be prepared, depending on the conditions involved.

The details of the actual construction of the plant are matters commonly in the category of millwrighting and will not be considered here.

The most common practice is to locate the mill for ore beneficiation near the mine. The reason is obvious: the mineral content is generally a very small percentage of the total ore; hence it saves money to get this mineral content out and the waste discarded as near the point of production as possible in order to save transportation charges on material that is going to be thrown over the dump eventually.

In general it is poor practice to locate a mill at the collar of the shaft, although this reduces to a minimum the surface transportation of the ore. But the fire hazard is great. If the mill should burn, the shaft is most likely to burn or be destroyed, and this may occasion loss of life underground, even though safety exits for mines are now required in practically all mining states. However, the closer the mill can be located to the shaft, the better, but the topography is an important and often a controlling factor. A suitable mill site cannot always be found without surface transportation of ore, which may be done by train, tramway, or aerial tram, depending on the circumstances and the situation. Besides the site for the mill itself, there must be ample room for waste and tailings disposal. In the case of the Utah Copper Co., with its tremendous tonnages mined and milled, it had to have tremendous areas for tailings disposal; hence the selection of its mill sites 19 miles from the mine. This is the extreme case in this country, and this problem declines, with the mill tonnage, to those few cases with small operations where there is substantially no problem at all.

In this age of low-grade ores, almost every mine has a mill and produces concentrates of one kind or another which are shipped to a smelter for further reduction.

It goes without saying that the plant must be located sufficiently far from the mining operation that any caving which results from mining, or from an accidental caving of a shaft, will have no effect upon the plant.

Whether the plant shall be of brick, wood, or steel, and the type of the construction, depends upon the life of the operation, the climate, the cost of material and construction, and the fire hazard. In Ontario, Canada, where extremely low temperatures throughout a long winter have to be provided against, plants that are expected to have a long life are more and more being built of brick. In the far north section of Ontario in the newly developed mining districts, however, wood-frame constructions covered with corrugated iron are the rule, but these are giving way in the larger operations to corrugated covered steel buildings. The cost of heating these structures through the winter season is an important item, and in some cases, in order to reduce that, the machinery has been crowded into a restricted floor area, greatly militating against convenience and efficiency. Proceeding southward across the United States and into Mexico, the plants become more and more open, depending on the climate, until plants are reached which consist merely of a covering to protect the machinery and the workmen from the sun.

Smelter Layout.—Thirty or forty years ago it was customary to consider the equipping of small base-metal mines with small smelters to make them independent of the so-called "Smelter Trust," and to save long hauls on ores. That plan proved to be unfeasible, and the day of small smelting plants built in narrow out-of-the-way canyons has probably passed. To make a profitable smelting enterprise, a large tonnage of ore must be available, together with the necessary flux, which requires

either a very large mine or a district producing a diversity of ores that will provide a suitable smelting fixture.

Moreover, large smelters may be equipped with bag houses, sulphuric acid plants, or other by-product plants, such as for the recovery of arsenic—economies impossible to the small plant.

There are large smelting plants in this country, operating with equal success, constructed on perfectly flat sites, and there are others constructed on sloping sites, thus giving an opportunity to bring the raw material in on the higher level and send it through the plant on a flow sheet of decreasing elevation, thereby saving the cost of raising large tonnages of ore from one treatment of it to another. Very much more ground space is required for a smelter than for a mill of equal capacity, and for cheap disposal of slag there should be as much drop in elevation at the slag dump as is possible. This is true whether the slag is disposed of molten or whether it is granulated.

Whereas a mill is built to serve one mine, a smelter is generally built to serve a group of mines and, therefore, there is much more latitude in choosing its location, and the main transportation lines are a much more important factor in determining such locations.

In a smelter there is one other problem to consider which does not arise in a mill, *viz.*, the disposal of smoke and fume. Smoke suits of very large proportions have handicapped many plants and, even when high stacks are provided, local atmospheric conditions often make stacks ineffective at times in disseminating the sulphurous acid gases so as to make them harmless to vegetation, if there happens to be farming or ranching, sometimes within a considerable number of miles. Therefore, the location of the smelter far away from any possible agricultural land is always desirable.

The mixing of ores for the copper-smelting charge may be done by running the charge train under the various bins pertaining to different kinds of ore, or a system of bedding may be used. The former is cheaper in initial outlay, but the ore bins, if constructed of wood, as they generally are, involve a considerable fire hazard and the cost of operations of the charge train is higher. When a bedding system is used, a considerable investment is required in structural steel and conveyers, but the operating cost is materially less.

A smelting plant requires very large quantities of water and power. The effects of these factors are not materially different from those involved in mills.

Location of Iron-smelting and Steel Plants.—With the gradual increase of population in this country from early times, the location of iron and steel plants has gradually moved westward almost in accordance with the westward shift in the center of population. There is an interesting reason for this that is not immediately apparent.

"The reasons generally assigned for the location of the iron industry have dealt chiefly with the question of whether the ore, of which about 2 tons are required per ton of iron, should be brought to the coal, of which about $1\frac{1}{2}$ tons are required, or vice versa. The coal is frequently coked before shipment and the weight of coke required is only 1 ton per ton of iron against nearly twice as much for the ore; but many plants have been located much closer to the coal than the ore, nevertheless. This fact has, not unnaturally troubled many who have attempted to work out the underlying causes.

"It may be freely admitted that the location of iron manufacturing centers is a matter of freight charges; but not, as is commonly assumed, of freight on raw material only. Obviously, there must be included freight on the finished product to the point of consumption. If we were to consider the establishment of an iron works in a country entirely without such works, knew that each person would consume a given weight of iron per year, and knew also the tonnage of the proposed works, we should know how much population was required to consume the product, and from the density

of the population in the country we could calculate the average distance the product would have to be hauled to reach the consumer, and the freight for this haul added to the freight on the raw materials would constitute the total freight charge on the product. It is this which should be kept a minimum. It would, clearly, be an accident if the point which gave the lowest combined value for the three were at either the ore mines or the coal mines."

Railroad freight rates are determined on the values of the product hauled. The greater the value of the product hauled, the higher is the freight rate charged. Inasmuch as the value of steel products is so much greater than the value per ton of the ingredients that are used in the manufacture of steel, the freight haul of the finished product to its market, the center of population, has a correspondingly greater influence in the determining factor. However, at present the development of the country is such that one single center of population is no longer the sole controlling center as was the case in the gradual trek of the steel plants from New Jersey to Pittsburgh and thence to Chicago. But the freight rate on the finished product to its market is still the important factor.

Iron and steel plants are almost invariably constructed in flat sites for two reasons: One is that the nature of the operations requires a flat site, and the other is that, being generally situated in nonmountainous country and a locality densely populated, other sites would with difficulty be obtainable even if desirable.

Estimating the Cost of a Plant.—A careful estimate of the cost of a plant must be made before any decision can be arrived at as to whether to construct or not. Such an estimate requires a preliminary survey of the ground, a preliminary layout of the plant, an estimate of the excavation required, and then a more or less detailed sketch of such a plant as is contemplated. Such a sketch will enable the engineer to calculate the quantities of timber or steel necessary in the construction. The cost of the mechanical units required in the plant are easily obtainable from the manufacturers. The other items of the cost become more and more detailed with the refinement of the estimate.

In estimating the cost of a plant, engineering judgment is of prime importance. The best engineers always add a large percentage for emergencies. All the items of an estimate should be calculated as closely as possible to actual cost and no allowances made. The allowance should be made finally in a lump sum, otherwise the final figure will be very misleading. The amount added varies with the enterprise, the various factors involved, and the personal equation of the engineer himself. If it is to be a plant that has many prototypes, a very close estimate of its probable cost can be made. If, however, it involves a new process and is the first plant of its kind to be constructed, the emergency factor must be more liberal. In some cases this has been known to be 30 or 50 per cent. Notwithstanding this allowance, very few plants are constructed within the estimated sum. However, here again great judgment and discretion must be shown, or the factor added will make the final sum so large as to discourage capital from undertaking the venture. The engineer who constructs the plant is rarely the engineer who operates it after it is built, and in such cases the former always comes in for a great deal of criticism, sometimes justly and sometimes unjustly, because of inconveniences that increase the cost of operation. Therefore, throughout the consideration and design of a plant, its utilitarian purpose must be borne in mind. It is something that must work efficiently if it is to justify its existence and do credit to its creators. It goes without saying, therefore, that operating experience is a *sine qua non* for the plant designer, and the earlier in his career he acquires it, the better off he will be.

CHAPTER XVII

THE ELECTRIC FURNACE IN NONFERROUS METALLURGY¹

Introduction.—The division between ferrous and nonferrous metallurgy is purely arbitrary, being suggested by economic convenience. It is not to be expected, therefore, that the principles governing electric-furnace practice in the one differ at all from those employed in the other. Furthermore, it should be clearly understood that there is no difference whatever in the fundamental functions of an electric and any other kind of metallurgical furnace, excepting only those cases in which the electric furnace serves the additional purpose of carrying on electrolysis.

The functions of an electric furnace are chiefly two: (1) as an energizing apparatus, by means of which the desired direction of reaction may be secured; (2) as a conditioning apparatus, by means of which a mixture may be reduced to the liquid or gaseous state for facilitating reactions and the mechanical control of the substances involved.

It is impossible to discuss intelligently the application, design, or construction of an electric furnace without referring constantly to certain basic factors and relations between electricity and heat. Electricity is only converted into heat when it is forced to overcome resistance. The classification of furnaces into arc-type, induction-type, resistance-type, etc., is useful for mechanical comparison, but it should be borne in mind that all electric furnaces are resistance furnaces, in the last analysis. Rodenhauser and Schoenawa, in their "Electric Furnaces in the Iron and Steel Industry," have very clearly briefed the fundamentals of electrical theory necessary for a clear understanding of electric-furnace design and operation. Much of the following is abstracted therefrom.

Ohm's law states: Current = drop in potential ÷ resistance, or

$$i = \frac{e}{r}$$

where i = current in amperes.

e = potential in volts.

r = resistance in ohms.

The resistance developed by a conductor may be expressed by the formula

$$r = \frac{cl}{q} = p \frac{l}{q}$$

where r = resistance in ohms.

c = the constant of the material.

p = c , or the specific resistance.

l = length of conductor in meters.

q = cross section of conductor in square millimeters.

The resistance of metallic conductors increases with rise in temperature, while the resistance of nonmetallic conductors, such as carbides, carbon, and refractories,

¹ This chapter is a major revision by the editor of the one contributed by H. H. Buckman to the first edition. Grateful acknowledgment is made of suggestions by the General Electric Co.'s engineering staff; O. Needham of Westinghouse Electric & Mfg. Co.; Dr. M. Tama of the Ajax-Wyatt Electric Furnace Co.; Dr. Victor Patchkis of Columbia University; W. E. Lewis and H. H. Watson of Pittsburgh Lectromelt Furnace Corp.; and R. L. Baldwin of National Carbon Co.

decreases with rise in temperature. This variation of resistance with the temperature is not regular, and is specific for each substance. Ordinarily speaking, variations in resistance for given differences in temperature are greater at higher than at lower temperatures.

Figure 1 illustrates the variation of temperature with resistance, according to Northrup, in magnetite. It may be considered typical of the so-called nonconductors.

For moderate temperature differences, the following formula is substantially correct:

$$r_t = r_0(1 + at + Bt^2)$$

where r_t is the resistance at t° ; r_0 the resistance at 0° ; a and B are numerical constants for each conductor.

The specific resistance p at 15°C . and the temperature coefficient a for a number of materials are given in Table 1.¹ The figures given are accurate only for

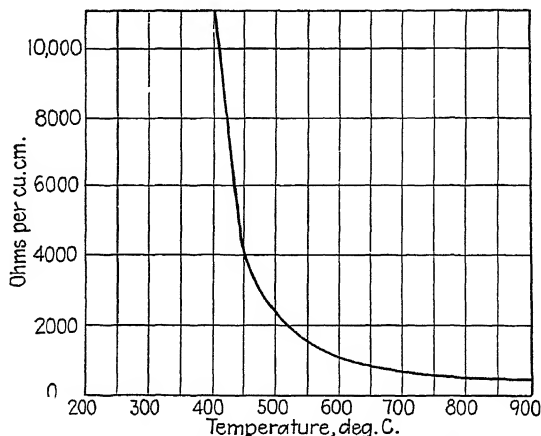


FIG. 1.—Variation of resistance in magnetite.

moderate temperatures. They cannot be used for accurate computations at high temperatures.

Series and Parallel Connections.—Both systems are employed in electric-furnace design and a clear conception of the fundamental factors governing the two is essential. Certain advantages and disadvantages accrue to each. Whether a furnace is parallel- or series-connected is usually dictated by the limitations of design in the particular work for which the furnace is intended. Figure 2 will serve to illustrate substantially all cases in both series- and parallel-connected furnaces. I in Fig. 2 represents schematically a furnace in which the current passes through one electrode, traverses the bath or charge, and then passes through the other electrode, in this way having to overcome the combined resistance of both electrodes and the bath or charge. Such an arrangement is designated as a series connection. It will be manifest that in this case, with a given voltage, the same amount of current will flow through the entire system and that the resistances of the electrodes and the bath are additive. On the other hand, II in Fig. 2 represents schematically an arrangement of the electrodes in

¹ Page 620. See also pp. 621-624.

parallel. Here the path of the current is divided. A portion will flow through one electrode, and a portion through the other, but both portions will traverse one-half the bath. The division of the path of the current in an electric-furnace system gives rise to some of the most difficult problems of design.

Kirchhoff's Laws.—Assuming that the ingoing currents are positive and the outgoing currents negative, Kirchhoff's first law may be stated: "At each point of division the sum of all the incoming currents equals the sum of the outgoing currents, or

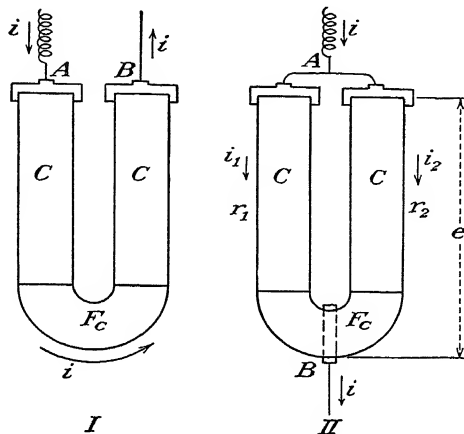


FIG. 2.—Series and parallel connections.¹

at each point of division the sum of all currents equals zero." From II in Fig. 2 it will be seen that

$$= \frac{e}{r_1}, \quad \text{and} \quad i_2 =$$

Hence

$$e = i_1 r_1 \text{ and } e = i_2 r_2$$

and

$$i_1 \times r_1 = i_2 \times r_2$$

and

$$i_1 : i_2 :: r_2 : r_1$$

This may be expressed verbally as follows: "Currents which flow parallel to each other vary inversely as the resistances of the parallel conductors."

Combined Resistances.—According to Kirchhoff's law, $i = i_1 + i_2$. But it is also true that $i = e/r$, $i_1 = e/r_1$, and $i_2 = e/r_2$.

Hence

$$\frac{e}{r} = \frac{e}{r_1} + \frac{e}{r_2} \text{ or } \frac{I}{r} = \frac{I}{r_1} + \frac{I}{r_2}$$

Hence the combined resistance:

$$r = \frac{r_1 \times r_2}{r_1 + r_2}$$

¹ RODENHAUSER and SCHOENAWA.

This holds for n parallel circuits

$$\frac{I}{r} = \frac{I}{r_1} + \frac{I}{r_2} + \frac{I}{r_3} + \dots + \frac{I}{r_n}$$

Conductivity Is the Reciprocal of Resistance.—Hence, in the case of n equal conductors

$$\frac{I}{r} = \frac{I}{r_1} + \frac{I}{r_1} + \frac{I}{r_1} + \dots + \frac{I}{r_1} = \frac{In}{r_1} \quad \text{or,} \quad r = \frac{r_1}{n}$$

or "The combined resistance of n parallel-connected equal resistances is equal to the n th part of any single resistance."

Any computations for actual furnace practice must allow for the actual specific resistance of both electrode and bath at *working temperatures*. In addition to this, every line of cleavage between two conducting media is potentially a source of extraordinarily high resistance, and these must be allowed for. For example, the resistance of the electrode at the point where an old stub has been screwed onto a new length will be the center of a high-resistance zone. The same is true for the point or zone of contact between the electrode clamp and the electrode, also the zone near the contact of the electrode with the bath or charge. If an arc is allowed to develop between the electrode tip and the bath or charge, the specific resistance of this arc must be taken into account. It is usually much greater than the total combined resistances of all other elements of the system. Under working conditions, practically all parts of the electric furnace become conductors, and their respective resistances have to be taken into account. The usual resistances directly in the circuit are as follows: (1) resistance of the transformer secondaries; (2) bus bars and leads; (3) electrode clamps; (4) zone between clamp and electrode; (5) electrode; (6) refractory surrounding the electrode; (7) arc, or zone of electrode contact with the bath or charge; (8) slag or slags; (9) bath proper; (10) lining; (11) furnace atmosphere.

The resistance of the metallic parts grows greater as the temperature rises, while that of the nonmetallic parts decreases. To accommodate these resistances in such a way that the mass of material necessarily used will be least and so that the greatest possible part of the heat developed by the resistance will enter the charge at the optimum temperature is the sum total of successful electric-furnace design and practice.

*Conversion of Electric Energy into Heat Energy Due to Resistance.*¹—Where an electric current flows through a conductor, a certain work is done. This work is expressed as *heat*. Upon this fundamental phenomenon is based all electric-furnace practice. So far as is known, electricity can be converted into heat only by overcoming resistance.

Joule's law states: "The heat developed by a current flowing through a conductor is directly proportional to the time, proportional to the resistance, and proportional to the square of the current."

$$Q = Ci^2rt$$

where Q = heat generated; t = time current flows; i = strength of current; r = resistance; C = constant dependent on the units chosen.

When the current is in amperes, the potential in volts, and the time in seconds, $C = 0.24$.

$$Q = 0.24i^2rt \text{ g.-cal.}$$

According to Ohm's law, $e = ir$. Hence

$$Q = 0.24eit \text{ g.-cal.}$$

¹ See Table 6, p. 625.

where e = volts; i = amperes; r = ohms; t = seconds.

Power (p) = ei watts = i^2r watts

$$1 \text{ kw.} = 101.98 \frac{\text{m.-kg.}}{\text{sec.}} = 1.360 \text{ hp.}$$

$$1 \text{ hp.} = 736 \text{ watts} = 75 \frac{\text{m.-kg.}}{\text{sec.}}$$

Delivered work (A) = eit watt-sec. or joules = i^2rt watt-sec. or joules.

$$1 \text{ watt-sec.} = 0.24 \text{ g.-cal.} = 1 \text{ joule} = 0.10198 \text{ m.-kg.}$$

$$1 \text{ watt-hr.} = 3600 \text{ joules} = 864.5 \text{ g.-cal.} = 367.1 \text{ m.-kg.}$$

$$1 \text{ kw.-hr.} = 1000 \text{ watt-hr.} = 864.5 \text{ kg.-cal.} = 367.114 \text{ m.-kg.}$$

$$1 \text{ m.-kg.} = 2.35 \text{ g.-cal.} = 9.806 \text{ watt-sec.}$$

Alternating Current.—Most electric furnaces are operated by alternating current, due to the undesirability of any electrolyzing action in all cases except those where the electrolysis of the metallic salt is essential to production, and also due to the great advantages in transmitting and transforming alternating current as compared with direct current, as well as the immense advantage of voltage control offered by induction transformers. Since the generation of alternating current is caused by the movement of a certain mass of copper alternately into and out of a magnetic field,

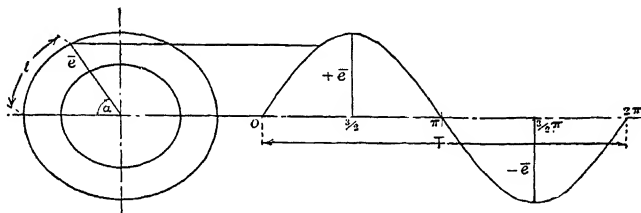


FIG. 3.—Sine curve, alternating current generator.

and since a certain interval of time is required for this generation, and, further, since one alternating current will generate and influence another in a near-by conductor, the flow of alternating current in conductors and the conversion of such current into heat are modified by certain factors which have to be taken into consideration in electric-furnace design.

Figure 3 illustrates the generation of alternating current, and is usually called the sine curve. The time required for one directional change, or the cycle, is represented by T . Let the radius vector equal the maximum voltage, \bar{e} , and the various instantaneous values on the curve be represented by e' .

Then $e' = \bar{e} \sin a$.

By substituting angular velocity for the angle $a = mt$ (distance = speed \times time) is obtained.

The angular velocity $w = da/dt$.

Hence $e' = \bar{e} \sin mt$.

Here t = time required for the radius vector to pass through the angle a . The corresponding angle for the time of the whole period, T , is 2π . Hence

$$2\pi = mT$$

Hence

$$m = \frac{2\pi}{T}$$

If v = cycles per second, or frequency, then

$$v = \frac{1}{T}$$

and

$$m = 2\pi v$$

Induction.—As the alternating current changes in strength continually and increases twice and decreases twice during each complete cycle, a conductor carrying such a current is surrounded by an alternating magnetic field which will induce countercurrents in all conductors within this field. Furthermore, the current-carrying conductor will induce in itself a counter e.m.f. due to this same cause. Both induction and self-induction are of primary importance in all alternating-current furnace design—especially in furnaces where the heating current itself is induced in the

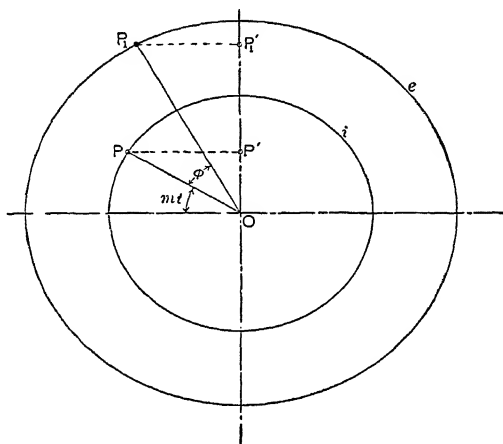


FIG. 4.—Current-voltage vector diagram.

resistor—and also in transmission and transformer design. The net effect of the self-induced current is to weaken the primary current. As the induction effect requires an appreciable time to develop, a current applied to a coil does not immediately reach its maximum. The current is said to lag behind the voltage.

If, as on the sine-curve diagram, the instantaneous values of the voltage can be represented by projections of a rotating radius vector, the instantaneous values of the current can be represented by projections of another radius vector of different value. The lag of the current behind the voltage can be drawn as a definite angle (see Fig. 4). This angle is the measure of the lag. The difference in time between the current and the voltage is called displacement. The angle between the radii vectors of the current and the voltage is the phase angle. The phase angle is usually denoted by the letter ϕ .

The self-induction developed in any conductor is dependent on the type of conductor and its position relative to other conductors. The factor expressing these conditions is called the "coefficient of self-induction" and is usually designated as L .

It will be seen that in alternating currents it is not only necessary to have an e.m.f. equal to the current multiplied by the resistance ($e = i \times r$), as with direct currents,

but an additional e.m.f. is necessary to overcome the resistance effect of self-induction. Hence, with alternating currents the total e.m.f. necessary is expressed thus

$$e = e_r + e_L$$

Here, e_L represents the e.m.f. of self-induction. Also,

$$\begin{aligned} e_L &= L \times \frac{di}{dt} \\ \frac{di}{dt} &= \frac{d\bar{i}(\sin mt)}{dt} = m\bar{i} \cos mt \\ e'_L &= (m \times \bar{i} \times L) \cos mt \end{aligned}$$

The formula for the total voltage is

$$e' = e'_r + e'_L = (\bar{i}r) \sin mt + (\bar{i}mL) \cos mt$$

Since $\cos mt = \sin (mt + 90^\circ)$, the voltage necessary to overcome the e.m.f. of self-induction is 90 deg. ahead of the e.m.f. necessary to overcome the ohmic resistance. Hence, these two e.m.f.'s must be added geometrically (see Fig. 5).

OA = maximum value of a current i .

OB = $e_r = \bar{i} \times r$.

OC = $e_L = \bar{i} mL$.

OD = the resultant of the two e.m.f.'s, as mentioned before.

$$e = e_r + e_L$$

Hence (Fig. 5)

$$\begin{aligned} &\sqrt{e_r^2 + e_L^2} \\ &\sqrt{\bar{i}^2 r^2 + \bar{i}^2 m^2 L^2} \\ &\bar{i} \sqrt{r^2 + m^2 L^2} \end{aligned}$$

Also $\tan \phi = mL/r$, if ϕ = the phase angle. It should be borne in mind that the values here discussed are the instantaneous and the maximum values. Usually, alternating-current instruments are built to indicate the so-called "effective values." Effective values may be obtained from the formulas here given by dividing the maximum values by $\sqrt{2}$

$$\begin{aligned} e &= \frac{\bar{e}}{\sqrt{2}} \\ i &= \frac{\bar{i}}{\sqrt{2}} \end{aligned}$$

Power Factor.—In direct-current circuits the power in watts may be expressed thus

$$p = e \times i$$

In alternating-current circuits, this equation must be written

$$p = e \times i \times \cos \phi$$

where e = effective voltage

i = effective current.

$\cos \phi$ = the power factor.

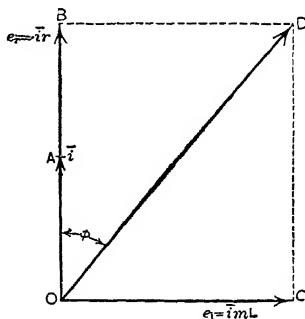


FIG. 5.—E.m.f. to overcome ohmic resistance and step inductance.

In alternating-current circuits the product ei is called the apparent power, and is expressed in volt-amperes. The *effective* power (wattage) is the apparent power modified by the $\cos \phi$, which is called the power factor.

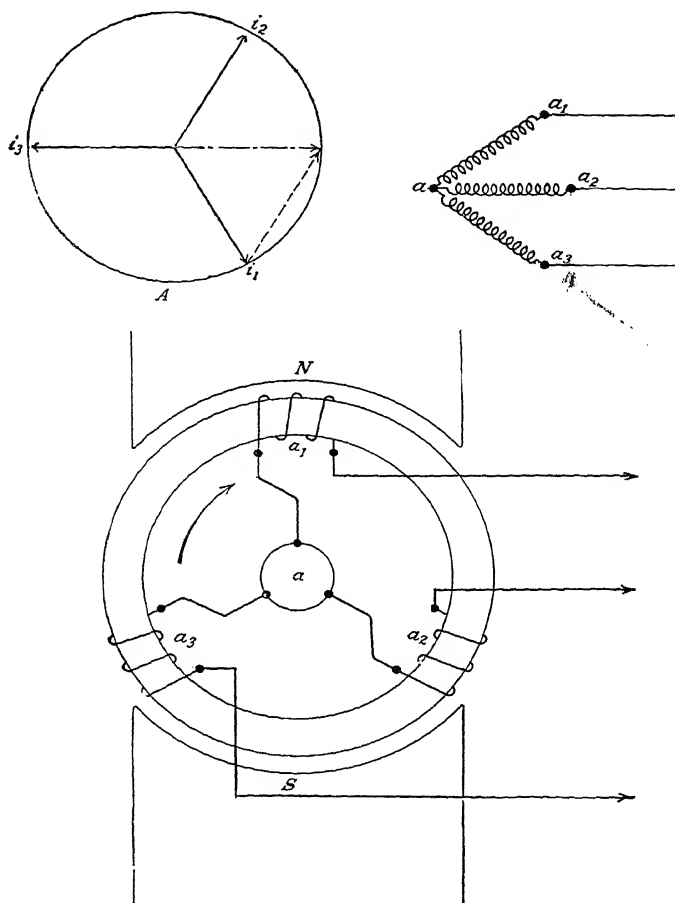


FIG. 6.—Star generator winding.¹

Where

$$\cos \phi = 1$$

$$p = ei \text{ and angle } \phi = 0^\circ$$

Where

$$\cos \phi = 0$$

$$p = 0 \text{ and angle } \phi = 90^\circ$$

¹ From RODENHAUSER and SCHOENAWA, "Electric Furnaces in the Iron and Steel Industry."

In all electric-furnace work it is necessary to carry comparatively high current densities on the conductors. The cross section of the conductors is determined by the current—hence the value of a high power factor that reduces the current. It should be remembered, however, that, while a low power factor means expensive conductors, it has no effect whatever on the prime mover.

Other Induction Phenomena.—In addition to the lag of the current, other induction effects have a direct bearing on electric-furnace design. In every metal part that is near an alternating-current conductor, there will be induced currents, with consequent loss. This is particularly the case in magnetic metal parts, such as iron or steel. In some cases these induced currents may cause dangerous or extremely wasteful temperatures in certain structural parts. In heavy currents, single conductors should

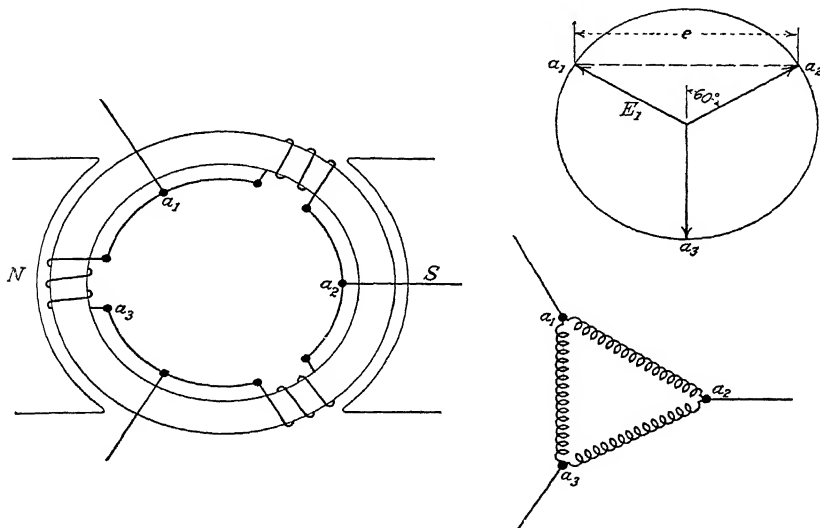


FIG. 7.—Delta system of generator winding.

never be allowed to parallel a steel beam at a short distance, or for any considerable length.

These induction effects may be largely neutralized by keeping the ingoing and outgoing conductors close together. Cooling chambers on electrodes should be made of copper, brass, or other nonmagnetic metal, to prevent the setting up of the so-called eddy or Foucault currents. Whenever a magnetic conductor of any considerable cross section entirely surrounds an alternating-current conductor, the constant demagnetizing influence of the current causes certain power losses. This phenomenon is called "hysteresis," and should be avoided in the design.

Polyphase Alternating Current.—For the purpose of this handbook it is probably sufficient to call attention to the fact that the foregoing discussion of alternating-current theory has been confined to single-phase currents, and merely to point out some of the outstanding characteristics of polyphase connections. The coils of these polyphase generators may be connected according to the system known as "star" (see Fig. 6) or that known as "delta" (see Fig. 7). In two- or three-phase work, two

or three currents, respectively, are actually handled. Figure 6A shows a vector diagram where the vectors are 120 deg. apart. Geometrically added, the resultant of any two of these current forces always equals the third. This is why only three conducting transmission lines are necessary. The third conductor may be regarded as a return for the other two, provided the current in the phases is balanced.

In the star winding the coils are connected to a neutral point at one end, while in the delta winding the coils are connected in series. With the star connection either the voltage of one phase or the resultant of two phases may be employed, *i.e.*, the end of one generator coil and the neutral point may be connected, or the end of one coil and the end of another. In star connections, different phase voltages make no difference in the current flowing through the coils or in the line. The generator phase current and the line current are equal.

In the delta arrangement, the phase voltage and the line voltages are equal, but the phase current and the line current are different.

Star connections are generally used with three-phase electric furnaces.

For star connections

$$e = 2E \sin 60^\circ = \sqrt{3} E =$$

For delta connections

$$i = \sqrt{3} I$$

For three-phase circuits, when the load on the phases is balanced,

$$p = 3EI \cos \phi = \sqrt{3} ei \cos \phi$$

When using a three-phase wattmeter to determine the total power on a three-phase circuit, and ammeters and voltmeters to indicate the current and the voltage of the three balanced phases, the power factor may be computed from the following equation:

$$\cos \phi = \frac{p}{3ei}$$

where p = total power (delta or star circuit); i = current in each line; e = voltage.

Electric Furnaces for Nonferrous Work.—The development of the electric furnace in nonferrous metallurgy has been considerably less extensive than in the manufacture of iron and steel, for two reasons: (1) the comparatively high melting point of iron, and its comparatively greater inertia toward oxygen at high temperatures, make possible the direct application of the carbon arc without too great oxidation losses; and (2) the value of the metal itself is so much less than those of the nonferrous metals that the oxidation losses sustained are less important in the economy of the metal. Electric furnaces are an expensive type of equipment, if circuit breakers, transformers, etc., are considered. They have therefore earned their place in the ferrous metal industry not because of cheapness of installation, but because of their ability to melt metal better than can be done with other types of equipment. Furthermore, the role of iron in the arts has resulted in the simple melting and refining operations occupying a greater proportion of the metallurgy of this metal than with the nonferrous metals. For these reasons the electric furnace is not yet so widely used in nonferrous metallurgy as in ferrous metallurgy.

The advantages of the electric furnace in nonferrous metallurgy are not economically so great as in iron or steel. Iron ore is sometimes located so that it cannot be economically smelted at all if it has to be transported to proper fuel, or vice versa. In

THE ELECTRIC FURNACE IN NONFERROUS METALLURGY

such cases it is sometimes possible to bring to the ore the energizing fuel, in the form of electricity. Here, the electric furnace has a fundamental advantage. On the other hand, such cases are much less frequent where the nonferrous ore is not of sufficient value to transport it to proper fuel, or proper fuel to it. Again, the high temperatures so quickly and easily obtained in the electric furnace are much more important in the metallurgy of iron and steel than in a majority of the more common nonferrous metals, which again tends to narrow the special field of the electric furnace in nonferrous metallurgy. This is not true in the border zone between nonferrous and ferrous metallurgy, where one is dealing with ferroalloys.

However, the electric furnace still holds many advantages over the fuel-fired furnace for many nonferrous metals. These advantages lie chiefly, although by no means exclusively, in the field of melting rather than of smelting. They may be listed as follows: (1) the obtaining of almost any desired temperature, with less duty laid on the refractory; (2) a temperature control superior to that in fuel-fired furnaces; (3) comparative freedom from contamination by noncarbon impurities in the fuel; (4) greater control of the furnace atmosphere; (5) more economical small units; (6) the avoidance of the necessity of heating great quantities of air; (7) the increased possibility of more convenient design, such as tilting and moving furnaces; (8) at least theoretically, a lower energy consumption; (9) easy reproducibility of operating conditions.

The disadvantages of the electric furnace may be listed as follows: (1) the high cost of electrically developed heat, compared with direct oxidation; (2) the high first cost of most installations as compared with fuel-fired installations; (3) trouble in operating, due to imperfect resistors; (4) the more frequent excessive temperatures; (5) the mechanical difficulties of constructing very large units; (6) the necessity for special and expensive refractory shapes, which are often very liable to breakage; (7) the rarely occurring suspension of current delivery during a heat.

Motor Effect of the Electric Current.—Currents of like direction attract each other, and currents of opposite direction repel each other. For this reason, in all fluid-metal masses, in both alternating- and direct-current furnaces, there is always a certain amount of motion not due to thermal causes. This is especially the case with arc furnaces, where the current is entering and leaving the bath on a line at right angles with the path it takes in traversing the bath.

The Pinch Effect.—The pinch effect is the name given to the contraction of molten conductors. When a molten conductor is subjected to a direct current, it appears to endeavor to contract its cross section. This is particularly noticeable in those portions of a metallic bath where there is already some slight constriction or shallowing of the hearth, which causes the current density automatically to increase, which, in turn, increases the pinch effect, and thus sometimes builds up this contraction of the cross section until the molten metal is actually pulled apart. As current densities in arc furnaces are relatively small, this effect is seldom conspicuous in this type. But in induction furnaces it is so marked as to be a distinct characteristic.

An explanation of the pinch effect is that the molten metal may be regarded as a bundle of an infinite number of conductors which, by virtue of the like directional currents flowing through them, tend to attract each other and, hence, to contract the mass. An actual breaking of the molten-metal mass, however, has only been observed in open-ring-type induction furnaces, which are no longer in use. In induction furnaces of the submerged-resistor type, the hydrostatic pressure, to which the metal is subject, prevents the rupture of the liquid metal contained in the resistor slots. Furthermore, if adequate current densities are used in this type of furnace, the pinch effect is beneficial in that the internal pressure created by virtue of the pinch effect is released at the places where the melting channels open into the hearth, thus cre-

ating a flow of metal out of the channels in the center portions with the corresponding return flow at the periphery.

Classification of Furnaces.—Many classifications of electric furnaces are in use. All are empirical and are arranged for convenience of the industry. The distinction between *direct-* and *alternating-current* furnaces needs no explanation. The *direct-current* furnace, except where electrolysis is desired, is seldom used. Except in electrolytic metallurgy, the alternating-current furnace is almost universally used in industrial work.

A common classification of furnaces divides them into three groups: (1) resistance furnaces; (2) arc furnaces; (3) induction furnaces. As all electric furnaces are *resistance* furnaces, this grouping merely indicates the kind of resistor employed, *i.e.*, whether (1) a solid or liquid through which a current is passed from one end to the other, or (2) an arc through which a current is passed, or (3) a ring or closed circuit through which an induced current is caused to flow. These three so-called types may be briefly described as follows.

Resistance Furnaces.—These furnaces are characterized by two chief types: (1) where the current passes through the material to be smelted or melted at a high current density and it is melted by the heat generated within the charge by its own resistance, and (2) where a special resistance material is employed through which the current passes, thereby heating it. This resistance material does not form a portion of the charge, but transfers its heat energy to the charge by direct radiation.

Arc Furnaces.—In arc furnaces, an arc is caused to develop in the furnace chamber. This arc transfers its heat to the bath either by radiation alone, or by both radiation and conduction. The arc may be generated between two electrodes out of contact with the charge, or between an electrode and the charge itself. Special cases are found where the electrode is made partially of the charge itself, and still others where the electrode is "buried" in the charge, giving rise to the so-called "submerged arc." In both these special cases the net result is a multiplicity of small arcs and considerable development of heat by direct resistance of the particles of the electrode and the charge.

Induction Furnaces.—The high-frequency induction furnace, which has become of great importance in nonferrous melting operations, is essentially a transformer without a core and operating usually at a high frequency. The coil is the primary of the transformer, and the secondary is the charge, which can be considered a coil of one turn short-circuited on itself.¹ While the effectiveness of this type of furnace depends on the supply frequency, on the resistivity, permeability, and the physical condition of the charge, on the coupling between the inducing coil and the charge, and on the heat insulation used, by far the most important single phenomenon involved in induction furnaces is the concentration of current on the surface of the charge, usually known as the "skin effect." The tendency of alternating current is to concentrate on the surface of the conductor. It requires a sensible time for the current to reach the inner layer of the conductor, and if the frequency is high enough, practically no current travels at the center of the conductor. Since the center of the conductor is of no practical value, hollow tubing is commonly used for the primary coil and cooling water is circulated through it to carry away the heat developed by the current in this coil and the heat conducted to the coil from the charge.

The higher the frequency used, the shallower is the concentration of the induced current. As a result of the current being confined to a comparatively small area, it induces a high local resistance and heats and melts this portion of the charge. The following table shows the effective depth of penetration with the various frequencies for different materials:¹

¹ CHESNUT, *The Westinghouse Eng.*, November, 1941.

DEPTH OF PENETRATION IN INCHES

Material	Frequency, cycles per sec.			
	60	1000	10,000	100,000
Graphite.....	10.1	2.5	0.78	0.25
Molten iron...	4.3	1.1	0.33	0.11
Stainless steel.	2.5	0.62	0.19	0.062
Molten copper	1.4	0.35	0.11	0.033
Copper.....	0.42	0.10	0.031	0.009
Cast iron.....	0.18	0.045	0.014	0.005
Rail steel.....	0.043	0.011	0.004	0.001

It may be noted, though this is more a matter of metalworking than of metal reduction or metal recovery, that it is possible to conduct various heat-treating and hardening operations through localized induction heating which cannot be carried on in any other way.

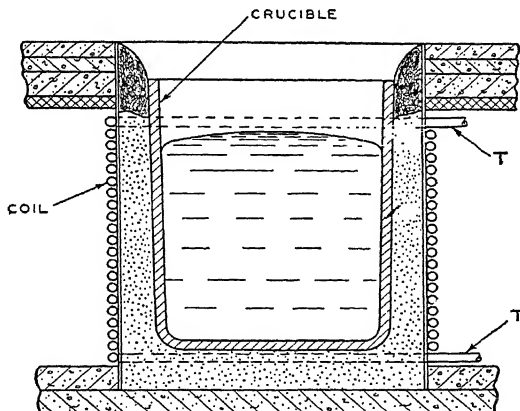


Fig. 8.—Induction furnace.

Adjacent conductors carrying current in the same direction attract each other and the charge can be considered as a number of concentric rings stacked on top of each other, each carrying current in the same direction and thereby attracting each other. A stirring action is set up in the charge, which is exceedingly helpful in certain reactions.

Since a conductor is necessary in the secondary of a transformer, if nonconducting materials are to be heated, it is necessary in this case to have a crucible that is a conductor. These are usually made of clay and graphite proportioned to give the desired electrical conductivity, temperature, and resistance to oxidation. The more graphite in the crucible, the higher is the conductivity and the permissible operating temperature, but the rate at which the crucible can oxidize is also increased.

The best method of controlling the stirring of the charge is to have currents of two frequencies in the same furnace coil. Low-frequency current can be used to stir the charge with little effect on the heating.

Since induction-furnace operation can be carried on in controlled atmosphere or even *in vacuo*, the induction furnace is usually operated without slag, although in such operations as the reduction of double beryllium halides by magnesium, the production of a scoria is an essential part of the operation. It should be remembered that the top of a charge in an induction furnace is not so hot as it is in a fuel-fired furnace. Any slags to be produced should therefore be calculated to have a low melting temperature.

Induction heating can of course be applied to autoclaves and large chemical vats such as are used in a bauxite industry. It is only necessary to make the shell of a magnetic material if such material is acted upon by the solution to be heated. A magnetic shell can be used with a liner of a resistant material.¹

There is also a low-frequency type of induction furnace which has attained a considerable vogue for melting brass, bronze, aluminum, and magnesium. In this

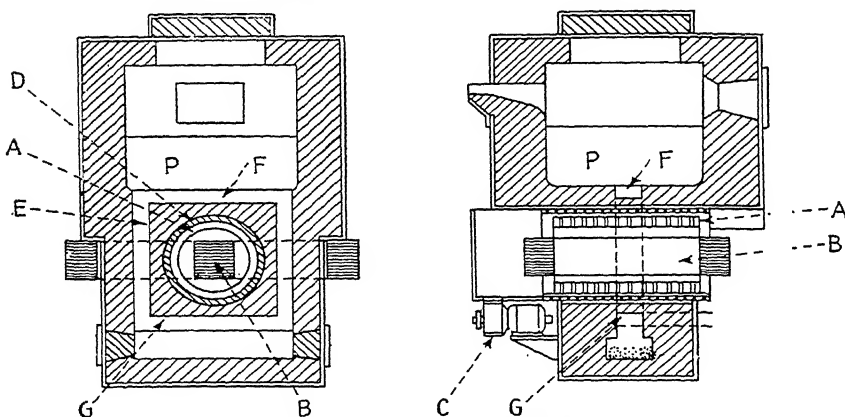


FIG. 9.—Induction furnace with melting channel.

furnace the primary coil instead of being around the crucible is arranged to be around a transformer core which is made of high-permeability silicon-steel sheets. There is a melting channel which is around the primary coil and not inside it, as in the high-frequency furnaces, and which itself becomes the secondary loop of the transformer. This melting channel opens into a hearth that contains the main charge. The heat is generated only in the melting channel, but because of the stirring effect which has already been spoken of, there is a constant flow of metal around this channel after melting has once begun which gives a very effective heating to the charge and rapidly melts all the material in the hearth.

A cross section of this furnace is also illustrated. The primary coil *A* is made of mica- or glass-tape-insulated copper strip arranged around the transformer core *B*. *D* is a transite insulating tube around the primary coil *A*. The blower *C* keeps the transformer winding and core cool during the operation. *EFG* is the melting channel which is the secondary loop and connects with the hearth *P*.

Between melts in the low-frequency induction furnace, it is necessary to keep reduced power on the furnace in order to prevent the metal in the stirring ring from freezing. The usual process is to have a step-down holding transformer for this

¹ CHESNUT, *The Westinghouse Eng.*, February, 1942.

purpose. Figure 10 shows a 250-kw. low-frequency Ajax-Tama-Wyatt induction furnace.

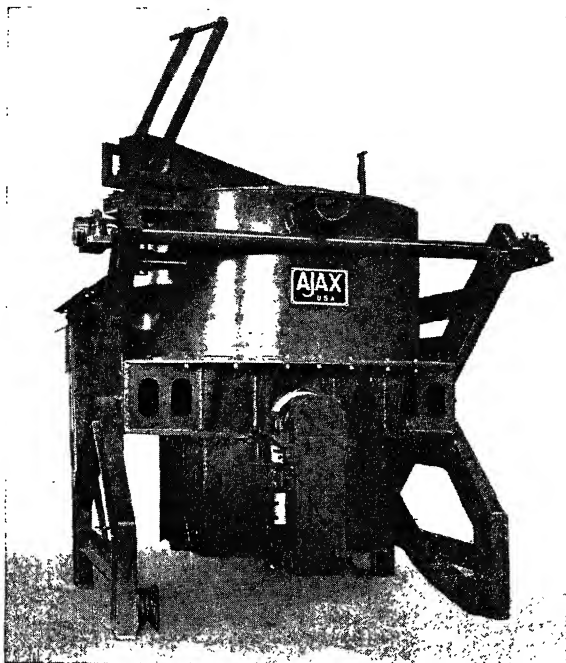


Fig. 10.—Low-frequency Ajax-Tama-Wyatt induction furnace.

The furnace atmosphere above the charge is heated only by the charge. This tends to cut down volatilization losses, but makes the use of any slag cover difficult. So far as the author knows, this type of furnace is not used in any smelting operations, but merely as a melting furnace.

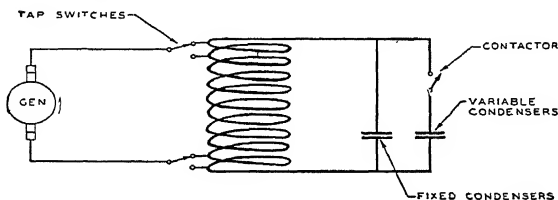


Fig. 11.—Capacitor diagram.

Capacitors.—In induction furnaces the power factor is low and decreases with the progress of a melt as the charge becomes molten which lowers the level and changes

the coupling of the furnace winding.¹ To improve the power factor and to avoid abnormally high field currents on the inductor alternator, which normally supplies the current, capacitors are introduced into the circuit. These are essentially large condensers, some of which are permanently connected with the circuit and others are switched in from time to time. The control equipment includes this switching means. Formerly, oil circuit breakers were used in these furnace circuits, but it has been

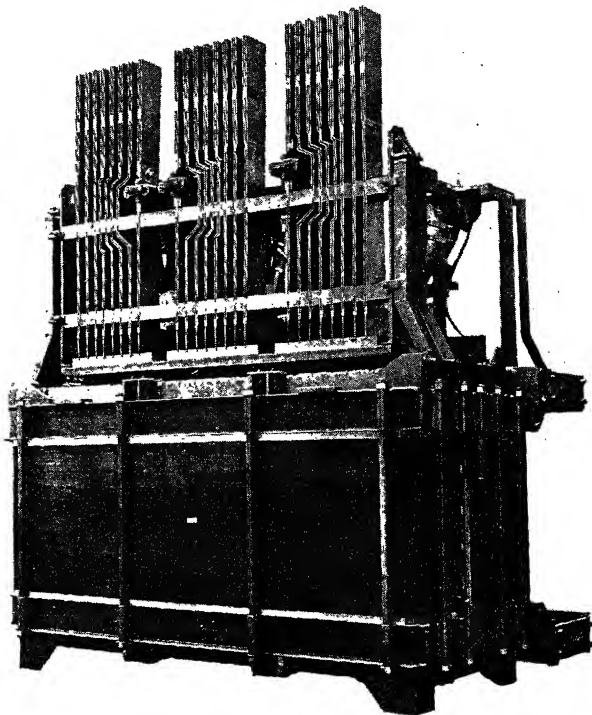


FIG. 12.—Westinghouse transformer and reactor.

found that contactors are more suitable than is a device that was designed primarily as a circuit breaker. Extensive descriptions of these capacitors does not seem necessary for this book as they should be the problem of the furnace supplier and should not be homemade by the metallurgist.

FURNACE TRANSFORMERS

Electric-furnace transformers are designed to "step down" the high power-line voltages to the lower voltages commonly used in furnace work, 90 to 210. Two usual characteristic features of such transformers are (1) the large amounts of copper in the

¹ ACKLEY, Control Equipment for Induction Melting and Heating Units, *Industrial Heating*, May, 1944, p. 716

secondaries to make possible the delivery of very heavy currents, and (2) a series of taps on the primary windings which enables the furnace operator to vary the voltage at will. A prime requisite of all such transformers is an ability to withstand very large overloads for short periods. They are made, of course, both single and polyphase.

The following description applies to Westinghouse standard electric-furnace transformers. The transformers are of the oil-insulated single-phase or polyphase type with self- or water-cooling, as local conditions may warrant. The shell form of construction is used as it is believed it permits better bracing of the windings to withstand surges.

The magnetic circuit is built up of silicon-steel sheets in order to reduce aging to a minimum. To reduce eddy currents, these sheets are annealed and given a core-plate varnish treatment. The laminated sheets are assembled in such a manner that they interlace at the corners, after which they are clamped between heavy steel frames. Provision is made for the circulation of oil around the built-up magnetic core.

The windings are made up of coils, subdivided into high-tension and low-tension groups. The arrangement of these groups is symmetrical. The spacing of each low-tension group from the neighboring high-tension group is made accurately and is the same throughout the windings, so that the impedances of the parallel low-tension coils are practically identical.

Furnace transformers differ somewhat from general power transformers. Practically all furnace transformers have six to eight taps with provision for changing the primary windings from delta to Y connections, some for full capacity and some for partial capacity. Different voltages may be applied to the charge, depending on the character of metal being melted and whether the furnace is in the melting or the refining stage. Higher voltages are used in the melting operations. Usually a no-load tap changer is included in the transformer case. The currents are comparatively high, and the secondary winding is heavy. Furnace transformers are usually oil-filled and water-cooled. The busses must be interwoven so that the ends of the secondary are brought out unconnected. However, the design of transformers is a problem for the manufacturer rather than for the metallurgist who uses them.

Reactance coils, or reactors, are almost always used in connection with arc-furnace circuits to prevent too heavy currents on short circuits. Such short circuits occur especially when "striking" the arc on a cold charge. Here the resistance of the charge is very low, and the conditions favorable to the establishment of a long arc, which would develop considerable resistance, are lacking. In such cases, the

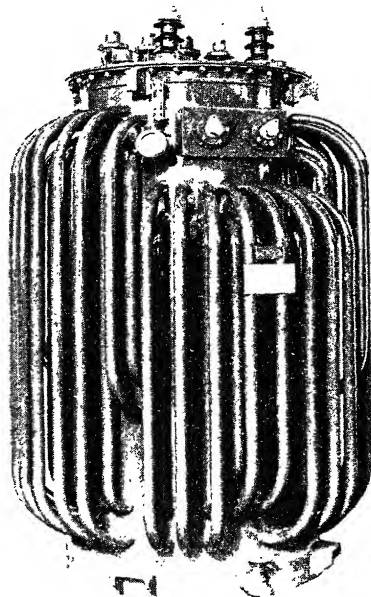


FIG. 13.—Kuhlman high-reactance transformer (with external voltage and reactance tap changers).

of current is momentarily very heavy on an unprotected circuit. The reactors are located on the high-tension side of the transformers, and are usually assembled in the transformer oil tank.

Regulation.—In true arc furnaces, where the bath has usually a comparatively low resistance, the arc is bound to be unstable. This instability is greater at low temperatures and in metals with a high boiling point. The arc tends to become more stable as the atmosphere above the bath becomes more and more charged with metallic vapor. But at best, the electrode end of the arc is in constant motion over the end of the electrode. As the resistance of the arc depends on its length (other factors being constant), it will be seen that, to maintain a constant energy input, rapid and positive voltage control will be required. The resistance of the arc may vary greatly in a space of a few seconds. There are several methods of control or regulation. The height of the electrode above the bath may be increased or decreased mechanically, which will increase or decrease the length of the arc. External resistance may be thrown into or out of the circuit. A third method is to place taps on the transformer windings and by this means to vary the voltage.

Regulators that govern by raising or lowering the electrodes are usually confined to arc furnaces. These regulators all operate by controlling the motors raising or lowering the electrode. The type of regulator that employs taps on the transformer coils is termed an "induction regulator." Many mechanical variations of both types have been placed on the market, and there has been a marked advance in the refinement of these machines in recent years. The cost of electric power to the furnace operator is largely based on demand, and it is essential, therefore, that the operation of the furnace be so regulated that high peaks in the load curve are avoided. Only automatic regulation can do this. Again, with automatic regulation, it is possible more quickly to bring a furnace charge to a temperature where furnace conditions are comparatively settled, and thus to reduce the total time for the heat, with a corresponding saving in radiation losses.

Modern controls are in the main based on the Ward-Leonard patents.

The Westinghouse regulator represents a modern apparatus adapted to arc-furnace work. This consists of two parts, the control panel and the regulator panel. The regulator panel carries the so-called control element, the necessary ordinary fuses and switches, as well as the magnetic switches controlling the electrode motors. On the back of this panel are placed the condensers and resistors in the control circuit. The so-called control element comprises a floating horizontal pivoted lever. At each end is a core extending into a solenoid. One of these solenoids is governed by the current flowing through the corresponding electrode. The other is governed by the voltage between the bath and the electrode. The pivoted arm has one contact on each side of the pivot, so arranged that magnetic contactors are operated by them. These magnetic contactors control the motors raising or lowering the electrode. When no current is flowing, this lever occupies a horizontal position. When the circuit is closed (the electrodes being out of contact with the charge), a potential exists between the charge and each electrode. This causes the voltage solenoid to pull its core and close the contacts at that end of the lever. This closes the contactors that control the electrode motor in the direction necessary to lower the electrode. Thus the electrode is brought into contact with the charge. At this moment the voltage coil will become deenergized as it is connected between the electrode and the furnace charge, and its current will be shunted. Thus the lever will return to the horizontal position and the motor circuit will be broken. When the second electrode makes contact with the charge and completes the circuit, the current solenoid is energized. This closes the contacts, which causes the motor to raise the electrode, which causes the generation of an arc between the electrode and the bath. As the electrode con-

tinues to rise, the voltage between the electrode and the charge increases, while the current decreases until the pull in the two solenoids is equal, which causes the lever again to come into the horizontal position, and this, in turn, stops the action of the motor. From this time on, whenever the arc fluctuates, it will cause the balance between the two solenoids to be overturned, and these will, in turn, cause the electrode motor to raise or lower the electrode and thus reestablish the balance. This type of regulator can be used at electrode speeds up to $2\frac{1}{2}$ ft. per min. and can be made sensitive to 5 per cent fluctuations in the current. Figure 14 shows schematically the arrangement of this regulator (connection for one electrode only) in connection with a three-phase arc furnace.

General Electric, Westinghouse, and Allis-Chalmers have recently introduced a new type of rotary regulator, known as Amplidyne, Rototrol, and Regulex, according to the manufacturer.

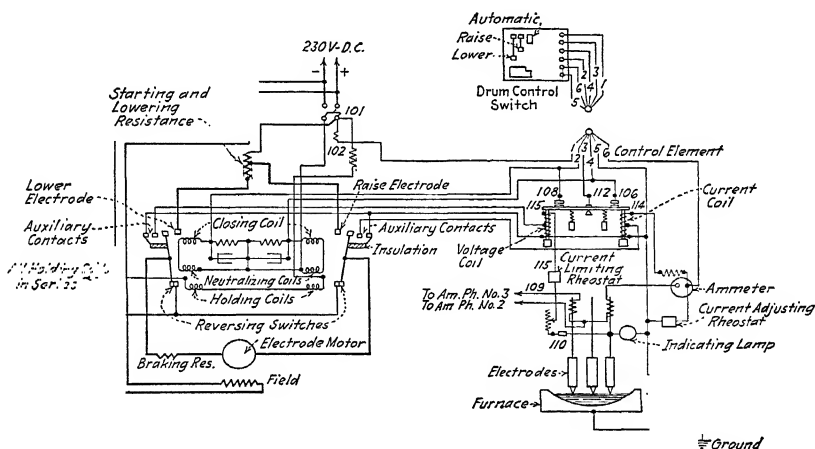


FIG. 14.—Westinghouse automatic current regulator.

Induction Regulators.—In furnaces of the direct-resistance or submerged-arc type (e.g., ore-smelting and carbide furnaces), when the electrodes are comparatively stationary, the current is not ordinarily subject to violent fluctuations. It usually gradually increases from the starting point up to the optimum rate of power input, where it is then desirable to control it and keep it steady. The usual method of securing this control is by regulating the voltage by a series of taps in the transformer high-voltage windings. The disadvantages of this method, while not serious, are often troublesome. If simple switches are used to connect with the several taps, the main circuit breaker must each time be thrown, causing a break in the arc. Furthermore, if the switches are not thrown in proper order, a portion of the transformer winding will be short-circuited.

In two-electrode single-phase or four-electrode two-phase furnaces, it is possible to have balanced phase currents, without having balanced voltages across the arc, for one electrode may be submerged in the steel.

With the rotary regulators, the voltages across the arcs are balanced, as well as the phase currents. One field of the rotary regulator is energized through a rectifier from a transformer in the electrode circuit, and a second is energized through

a rectifier, from the voltage across the electrode to the shell of the furnace. It may be noted that because of the utilization of this voltage difference electrical contact must be made in some way between the furnace shell and the furnace lining. In basic-lined furnaces the lining itself will provide such a current, but in an acid-lined furnace, either metal electrodes must extend through the lining, or a scrap-steel bar may be inserted into the charge and the furnace door clamped down on it.¹

There seems to be some saving in power through the use of the rotary regulator, but there is a very decided saving in electrodes.

Electronic Control.—A new control that appears to be faster in response than any other type is based on electronics and was described in *A.I.E.E. Technology Paper* 44-164, May, 1944, but no extended results are available.

Furnace Conductors and Connections.—The salient feature in an electric-furnace installation is the current density in the secondary conductors. This may often reach

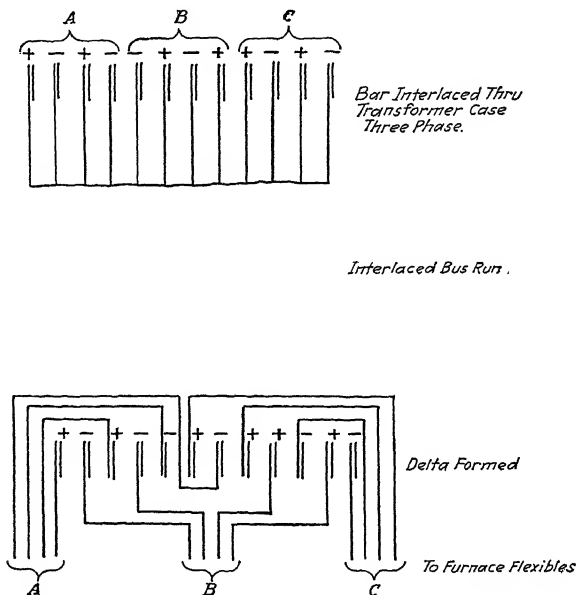


FIG. 15.—Bus bars interlacing to reduce inductance losses.

many thousand amperes. As alternating current is almost exclusively used, the furnace designer is confronted with two difficult factors which do not have to be considered in direct-current transmission. The first of these is the so-called "skin effect," and the second is inductance.

Skin effect is the term applied to the tendency of an alternating current to seek the surface of a conductor. This has already been discussed under induction furnaces on page 594, and, as has also been pointed out there, it is possible to remove a portion of a cylindrical conductor, with little effect on its conductivity. Advantage is taken of this by making heavy conductors tubular in shape, or by building up heavy coils on a nonconducting core.

¹ "Westinghouse Descriptive Data," 31-280, March, 1944.

The two essential features to be provided in a set of furnace conductors therefore are (1) an arrangement of the individual bus bars or leads which will result in the minimum inductance effect, and (2) to provide the leads with cross sections properly shaped and adequate in size to prevent excessive heating.

Obviously, that conductor which has the greatest surface for a given cross section will show least loss of conductivity due to skin effect. Therefore, the worst cross section is a solid cylinder and the best is that of a thin sheet. In practice, the thin sheet is usually bent in the form of a tube. This results not only in lowering the losses due to skin effect, but also it assists in cooling, and hence lowers the losses due to heating. Where tubular conductors are not practicable, flat bars are to be preferred.

To avoid inductance losses, the bus bars must be so arranged that the inductance due to the alternating-current flow will be least. As the current in a conductor of given polarity tends to flow near the surface adjacent to a conductor of opposite polarity, it is best to alternate or interlace bus bars and leads as far as possible.

Figure 15 shows the furnace bus bars interlaced in the same manner as the transformer leads.

In three-phase connections where there are three-phase groups of leads, the two outer groups will necessarily have a greater inductance than the center group. To

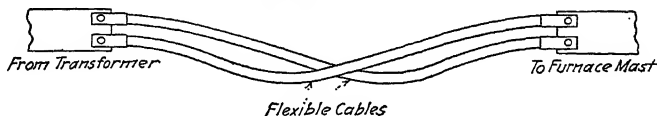


FIG. 15a.—Cable transposition.

overcome this, transposition is resorted to when it is feasible, in order to give equal impedance and hence equal current to each cable, thus avoiding overheating. Figure 15a shows a simple case of transposition in flexible cables.

Connections.—In heavy furnace-conductor connections, the usual form of soldered joints are not feasible, owing to the possibility of heating and melting of the joint. For heavy currents, too much care cannot be taken in making these connections, as the tendency to heat is always present. Joints that have functioned perfectly for a long time will suddenly and without apparent cause begin to heat violently. This is usually due to a former heating caused by overload and the consequent formation of a thin film of oxide on portions of the joint not airtight. After a joint is made mechanically sound and clean and is bolted tight, it will pay to see that the line is never sufficiently overloaded to cause undue heating, for this results in expansion and subsequent contraction at the joint. Where the bolts have a different coefficient of expansion from the bars, oxidation is sure to result ultimately, with subsequent disastrous heating. Naked copper conductors should be used whenever possible, except in the flexible furnace leads themselves, and even here the use of asbestos-covered cables is open to question, although they are undoubtedly safer for the workmen.

Resistor Connections.—Under this head should be grouped that portion of the circuit which is between the end of the flexible cable leading to the furnace and the resistor itself, whether the resistor is a solid, a liquid, or a granulated mass. It includes electrode clamps. This is the most difficult portion of the circuit to maintain outside the resistor itself. The difficulty arises from the high temperature and the contact resistance between two different substances. In the case of carbon-electrodes the clamps are almost always of copper and are often water cooled. Water cooling of the clamp has the great advantage of preventing oxidation of the copper and consequent heating. The danger of this heating lies in the fact that it is liable to become progressive. With oxidation comes an uneven distribution of current at the actual point of

contact, resulting in intense local heating and local destruction of contact, which is likely to develop small arcs. Once these develop, destruction of the clamp itself is certain unless the current is immediately suspended.

It is usual to support the electrode clamp with the electrode holder, which is made of steel. The function of this latter is to hold the electrode in place and to afford a movable support by means of which the electrode may be thrust into or withdrawn from the hearth or arcing zone. When cooling devices are resorted to, they are also carried on the electrode holder. Of course, the metal portions of the holder must be insulated from the copper pieces that form the terminals of the flexible cables and that form the electrode clamp. This clamp should have an area of contact with the electrode at least three times the computed area necessary to carry the desired current, in order to overcome the contact resistance. On the smallest size electrodes they grip the electrode over practically its entire circumference and are bolted in such a way as to ensure an even pressure over the entire contact surface. On large furnaces a wedge is used, either manually or power operated, to squeeze the electrode against a contact plate.

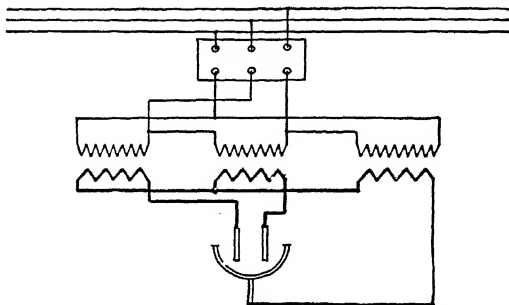


FIG. 16.—Three-phase connection, star-connected secondary, hearth forming one electrode.

Electrode Hearth or Bottom Connections.—This is a special case of connection where the connector must bridge the gap between the flexible lead and the molten material of the hearth. It is used in all cases where the bath itself forms one electrode. Two general methods are employed to make this connection: (1) There is inserted into the built-up hearth bottom a metallic electrode, provided with a water-cooling device which keeps its lower end from melting. (2) The bottom electrode is made of metal, but it is tapered with the small end toward the bottom, so that the lower portion always remains unmelted. In both cases, the electrode must be of the same metal that is being treated in the hearth, as the top end will always be melted in practice. In these cases, also, it is always necessary to leave enough metal in the furnace after pouring or tapping to constitute the bottom electrode for the next heat. The bottom end of the electrode may be brought out through the furnace shell through an insulated duct and clamped to the copper terminal of the flexible lead on the outside, or it may be bolted to the steel shell of the furnace from the inside, in which case connection is made by bolting the flexible lead terminal direct to the shell at any convenient point.

Both methods have been used, but, in general, it is true that the simplest possible construction is the best and that furnace hearths are never free from liability to cracks, which at best makes the introduction of cooling water a potential source of danger from explosions due to steam generated by accidental contact of molten metal and the water. It is generally the best policy to have no openings of any kind in the under

side of the furnace shell, for if the refractory hearth fails ever so slightly, molten metal is sure to escape and to cause loss and possibly serious damage. The use of bottom electrodes to secure greater temperature uniformity of the charge is attractive in

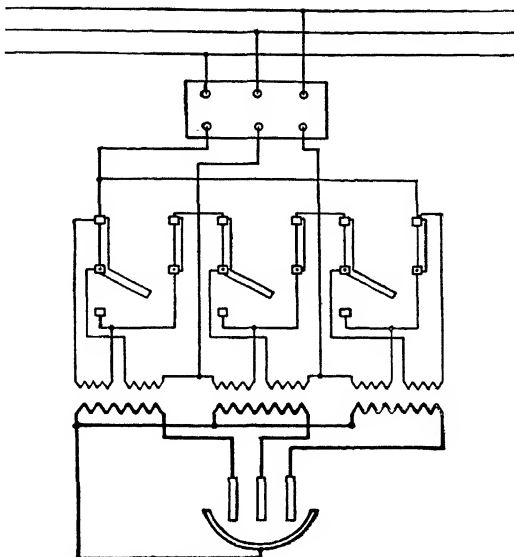


FIG. 17.—Three-phase connection—star secondary—delta primary.

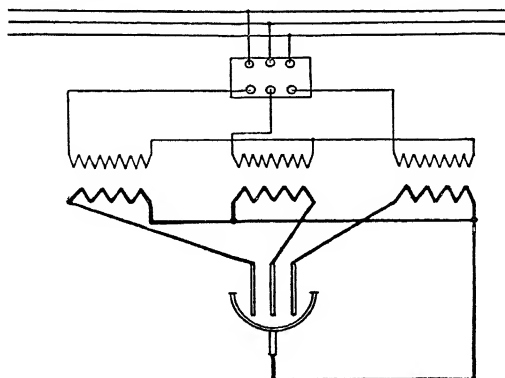


FIG. 18.—Three-phase connection—one phase reversed. Practically all of current goes through bottom electrode.

theory, but does not work out so well in practice, as most of the heat is generated in the arc.

Connection between Flexible-cable Terminal and a Solid-carbon or Graphite Resistor. Here the connection should be made outside the furnace, i.e., the carbon resistor

should project far enough beyond the actual furnace chamber to be kept cool enough by simple radiation to prevent oxidation of the copper connecting clamp. This clamp should provide sufficient heat capacity and radiating surface to keep the joint from overheating. This type of joint, *i.e.*, between the copper flexible lead and an incandescent carbon or graphite resistor, is most difficult to maintain, and is still one of the weak points in this type of furnace, although much progress has been made in this respect.

Connection between Flexible-cable Terminal and a Granulated Resistor.—This is usually made by the same type of connection as described in the last paragraph, where a carbon block or rod extends from the copper clamp on the cable terminal through the wall of the furnace, and is packed in the resistor material. This is also a troublesome joint to maintain, and the perfect detail has yet to be worked out. Slow

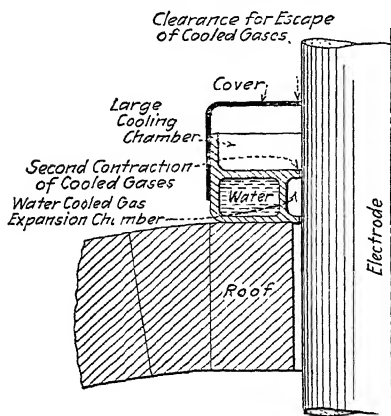


FIG. 19.—Furnace roof with electrode economizer.¹

oxidation of the connecting carbon block or rod (usually called the contact block) and the consequent reduction in cross section is followed by increased resistance to the current. This causes still greater heating and more oxidation. The consequence is rather rapid destruction of the contact piece and failure of the connection. This fault can be only partially overcome by increasing the size of the contactor block. The author has conducted many experiments with numerous coatings that might be applied to such contact pieces and prevent their oxidation, but with meager success.

Resistors.—Any portion of an electro-thermic machine that contributes appreciably to the total heat of the system because of its ohmic resistance may be regarded as a resistor. Whether this contribution is accidental and unintentional, especially arranged for, or merely unavoidable is immaterial. It is usual to speak of certain resistors as *electrodes*, the general meaning attached to which is that their primary purpose is to carry the current to the principal resistor whose primary function is the development of heat. Thus, in arc furnaces, the primary resistor is the arc itself and the chief function of the electrode is to conduct the current to the arc. Again, in resistance furnaces, the chief function of the electrode is to carry the current from the bus bar or flexible lead to the inside of the furnace and the primary resistor. In both cases, however, there is always a certain amount of heat generated in the electrode itself, and often this is an important function of the electrode. It is, therefore, permissible to class electrodes among the resistors. Many parts of the electric furnace function as resistors and contribute heat to the system. For practical purposes, however, the following classification is sufficient: carbon and graphite electrodes, solid-carbon or graphite resistors, granular-carbon or graphite resistors, metallic resistors, nonmetallic resistors, slag resistors, gaseous resistors.

Carbon Electrodes.—These are made from petroleum coke and selected anthracite coal, by grinding the carbon, mixing it with coal-tar pitch, molding or extruding under high pressure, and then baking.

¹ See Tables 10 to 11, p. 6280

The National Carbon Co. gives the following specifications for electrodes for electrothermic work:

Trueness to Form.—1. No electrode shall vary more than ± 2 per cent of the diameter from the normal diameter. For instance, a 17-in. electrode shall not be over 17.34 in. in diameter nor under 16.66 in.

2. The curvature shall not exceed 0.5 per cent of the length measured by the chord-to-arc method.

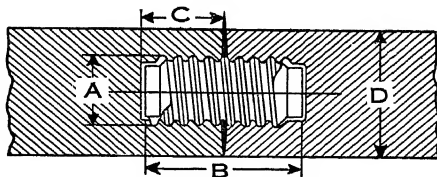


FIG. 20.—Standard carbon nipple, round top threads.

3. The ends of the electrode are to be straight and square with the sides; that is, a straightedge laid on the end of the electrode shall be 90 deg. from a straightedge laid on the side of the electrode for a distance of 9 in. from the end referred to.

4. Every threaded section shall be centered, and at no time shall the overhang to adjacent sections extend more than is permissible under the diameter tolerance.

Apparent Density.—The apparent density shall not be less than 1.50 in all sizes of coal electrodes.



FIG. 21.—Graphite electrodes. (Courtesy of National Carbon Co.)

Resistivity.—The specific resistance of no electrode shall exceed 0.0025 ohms per inch cube at 20°C.

Chemical Composition.—The ash shall not exceed $8\frac{1}{2}$ per cent.

Resistor Carbon and Graphite.—Standard sizes of National Carbon Co. carbon resistors are as follows: $\frac{1}{4}$ to $\frac{1}{8}$ in.; $\frac{1}{8}$ to $\frac{1}{64}$ in.; $\frac{3}{16}$ in. to dust.

Graphite resistor is carried screened as follows:

93 per cent through 3-mesh on 8-mesh
90 per cent through 8-mesh on 20-mesh
98 per cent through 20-mesh
50 per cent through 35-mesh on 65-mesh
80 per cent through 8-mesh on 20-mesh

Acheson Graphite Electrodes.¹—These are made of petroleum coke particles, mixed with a binder, extruded, baked in a gas-fired furnace, and then heated in an electric furnace to a temperature necessary to convert the amorphous carbon into graphite. Acheson graphite electrodes are chiefly distinguished from amorphous carbon electrodes by their greater conductivity and the greater ease with which they may be machined. Whether the graphite or the amorphous carbon electrode is superior is still an unanswered question. Each has certain points of superiority over the other, and entirely successful practice has been achieved with both.

Söderberg Electrodes.—The Söderberg electrode is self-baking and is continuously formed. The mixture is carbon mixed with more tar or pitch than is used in the ordinary electrode, and the mixture is fed into a metal cylinder which is above the electric-furnace charge. For ferrous alloys, steel, and carbides, the cylinder is usually made of sheet iron 1 mm. thick and perforated with a number of small holes to allow the evolved gases to escape. For use in the aluminum furnaces, it is made of thin aluminum sheet. The casing is of course eaten away as the electrode is used up. New sheet-metal sections are welded on as needed at the top of the electrode, which can be of any desired cross section. The only difficulty in the use of the Söderberg electrodes is that when a furnace is being started the electrode may be consumed faster than it is properly baked owing to the fact that one has a comparatively cold furnace and at the same time a high electrode consumption. The use of the Söderberg electrodes in aluminum manufacture allows a much better hooding of the furnaces so that practically all the fluorine that is evolved during the electrolysis can be recovered. The evolved gases are scrubbed with a caustic-soda solution, producing sodium fluoride, which is used in the manufacture of synthetic cryolite, which is then returned to the process.

The Söderberg continuous zinc oxide furnace is an application of the Söderberg electrode to the production of zinc oxide direct from the ore instead of from spelter. The process is a trade secret, but it consists essentially of making Söderberg electrodes of zinc ore, and then passing a heavy current through them, at the same time providing an oxidizing atmosphere. The zinc oxide volatilizes off.

Another self-baking electrode for aluminum manufacture is that of George Hagerup-Larssen (U.S. Patent 2,238,936), in which plates are used above the point where baking begins, but in the flow of the stream of electrode-forming material, in order to ensure proper flow of the mix.

Solid Amorphous Carbon Resistors.²—When radiation is unimpeded, a carbon rod through which is passed a wattage equal to 100 watts for every square inch of radiation surface will develop a temperature of approximately 1100°C. To attain a temperature of 2450°C., the wattage per square inch must be increased to 1,600.

With a current of 300 amp. and e.m.f. of 50 volts, the following lengths of carbon rod are needed in the respective sizes to develop a temperature of 1100°C.:

¹ National Carbon Co.

² See Table 11, p. 628.

Diameter, Inches	Length, Inches
1½	32
1¼	38
1	48
¾	72
½	95

Solid Metallic Resistors.—This type of resistor is generally used for furnace temperatures of 2000°F. and under. Like all solid resistors, the temperature limits of metallic resistors are set by melting point and oxidation speed. So far as melting point is concerned, metallic resistors are decidedly inferior to carbon, but carbon at high temperatures is rapidly oxidized. The furnace designer is caught on one horn of the dilemma or the other. The metallic resistor for comparatively low-temperature work has many advantages. With proper alloys, it is possible to get both strength and very considerable resistance to oxidation. This makes possible more or less elaborate design (often quite essential for even distribution of heat through the furnace) and a reasonable life for the furnace.

Of course, after oxidation has progressed to a certain degree, the resistor is doomed to destruction, for at higher temperatures oxidation will increase at an accelerated rate, while the development of heat in the resistor depends on the cross section which is being lessened at an accelerated rate by oxidation. Consequently, once a metallic resistor begins to oxidize it is liable to "burn out" very rapidly owing to the heating of the reduced cross section to melting point and the formation of an arc of still higher temperature at the point of rupture. So far, the choice of metals suitable for solid resistors has been extremely limited. Iron cannot be successfully used even for moderate-temperature furnaces, because of the tendency to oxidize rapidly. It is also especially difficult to maintain an unoxidized joint. The same is true of copper and nickel. Silicon has too little strength. Most other metals have too low melting points, or are too expensive. Practically, the only commercially available metals for such purposes are certain alloys of chromium and nickel. Some of these have been patented under the trade names of "Chromel," "Nichrome," etc. Though these alloys leave much to be desired, they combine to a sufficiently advantageous degree specific resistance, low oxidation tendency, and comparatively high melting point to make possible the construction of many useful and fairly efficient furnaces for low-temperature work. It may be noted that the Hevi-Duty Furnace Co. believes that it has an alloy that will give nearly a year's service at 2300°F.

Nonmetallic Resistors.—Solid and granular carbons have long been the mainstay of the metallurgist. An accompanying illustration gives the results of some recent work of the staff of the National Carbon Co. on resistivities of various forms of carbon at various temperatures. The carbon curves are carried only to 1000°C., as graphitization begins at about this temperature, and the curves begin to approach and finally merge in the graphite curves (see page 610).

In nonmetallic resistors the so-called Globar (silicon carbide) nonmetallic heating element represents a great comparatively recent advance. The Harper Electric Furnace Co. of Niagara Falls builds furnaces with these elements for steady operating temperatures of 2500°F., and maximum temperatures of 2750°F.¹

Slag Resistors.—Slags are used for resistors in metallurgical work, chiefly in smelting and refining work. Like all conductors of the second order, ordinary slags must reach a fairly high temperature before they will conduct an appreciable amount

¹ *Industrial Heating*, September, 1944, p. 1532.

of current. In refining furnaces, the slag usually is out of contact with the carbon electrodes, but serves as an intermediate resistor between the arc of one electrode and the metallic bath. In such cases, the resistance of the slag is not the main source of heat, but nevertheless contributes substantially to the total furnace heat. On the other hand, in smelting or reduction furnaces, the electrode is often thrust into the molten slag, and the ohmic resistance of the slag then becomes the chief source of heat.

Gaseous Resistors.—These are the arcs. It is not within the province of this work to go into the extensive field of electric-arc phenomena. A few fundamental

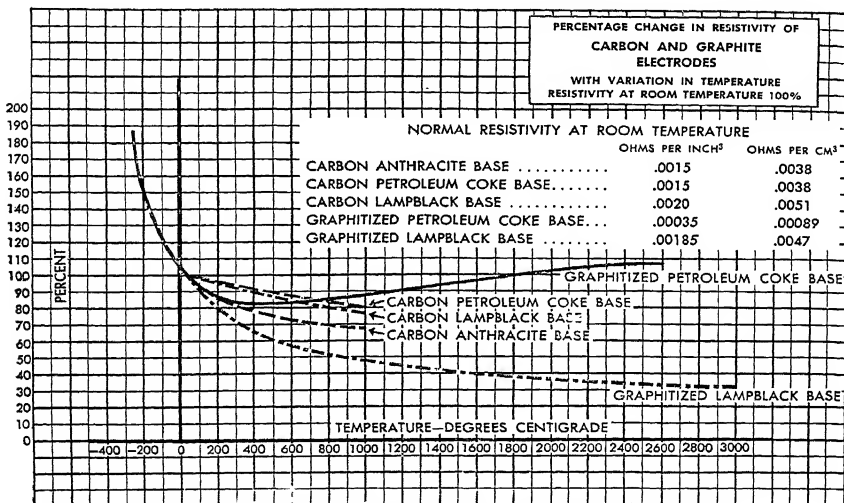


Fig. 22.—Resistivity curves, carbon and graphite electrodes.

considerations will be of help, however, in understanding arc furnaces. The heat developed by the arc may be regarded for practical purposes as due to the resistance of a vapor or a mixture of vapors. The true carbon arc is known only in the laboratory.

The Pointing of Electrodes.—Regardless of their shape when first placed in operation, carbon or graphite electrodes will always tend to develop a tapering point after a few hours of arcing. This is illustrated in Fig. 23. The pointing process will proceed to a degree where the inward feed of the electrode is equal to speed of the wearing-away process, after which the point will maintain more or less a roughly fixed outline. This pointing of the electrode is simply an expression of the relation between the arcing and oxidation tendencies on the one hand, and the temperature gradient in the electrode from the arc outward on the other. A close examination of an electrode at high temperature and with the arc in play (made by means of heavy cobalt glass) will show not only the results of the invisible oxidizing reaction, but also a visible wearing of the electrode, tiny incandescent particles being projected away from the electrode radially and in swarms. The net result is a greater wearing away of the electrode at the hotter end and a gradually lessened amount as the cooler end is approached, resulting in the tapered or pointed shape.

Industrial Melting and Smelting Furnaces.—A great variety of nonferrous melting furnaces have been placed on the market and are in daily use. They involve both the arc and solid-resistor types, as well as the so-called "induction furnaces." A few types are mentioned below.

Héroult Type Electric Furnace.—The Héroult furnace is one of the oldest of the electric furnaces, being first used commercially about 1905. It is an arc furnace with the arcs being struck between the electrodes and the charge, top electrodes furnishing the conductors for both the incoming and outgoing current. The original furnace was intended as a steel melting furnace, and in order to avoid an undue carburizing of the charge Héroult soon evolved the idea of covering the charge with a protecting layer of slag. The arcs were then struck between the electrodes and the slag, and the electrodes were far enough apart and the slag layer was sufficiently thin for the current to pass through the slag and through the metal underneath it rather than to take a

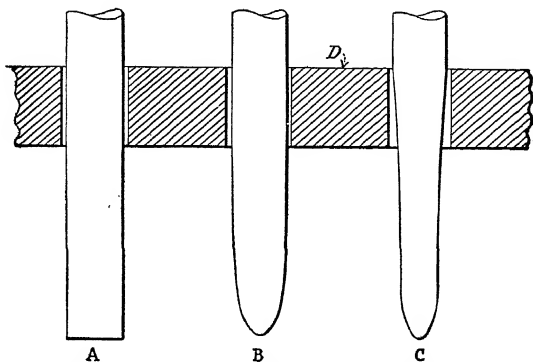


FIG. 23.—A, new electrode; B, normal point from spalling and oxidation; C, extreme pointing due to excessive hot-gas escape.

short cut by arcing from one electrode to the other. The furnace is still in use and has a considerable vogue for melting the more refractory nonferrous alloys. Figure 24 shows a modern Héroult furnace for nonferrous work taking a $7\frac{1}{2}$ -ton charge.

Probably the largest nonferrous electric smelting furnace is that designed by Jens Westley, which was in use before the war at Outokumpu, Finland. This was a three-phase 2500-kva. furnace, 10 meters outside diameter for smelting 240 to 250 tons of cold charge per day. The current consumption was 500 to 600 kw.-hr. per ton of ore smelted, and the electrode consumption was 2 to 3 kg. per ton of ore. The electrodes (three in number) were of the Söderberg type, were each 1.4 meters in diameter, and each weighed 5 tons.

The furnace was charged mechanically from overhead hoppers, the feed running about 25 per cent copper, producing a 50 per cent and a slag run 0.3 to 0.6 per cent copper. The slag flow was continuous, the matte flow intermittent. The converter slag went back to the furnace as a flux.

An interesting feature of this installation was the recovery of sulphur dioxide from the waste gases. These were washed with a basic aluminum-sulphate solution which was kept in circuit until it contained about 70 g. of sulphur dioxide per liter. This was then recovered by heating the scrubber solution to $100^{\circ}\text{C}.$ ¹

¹ *S.M. Eng. Min. Jour.*, Vol. 140, p. 47, January, 1939.

The Bassanese furnace (Stassano) is important historically. It was a two-electrode arc-type furnace, the most distinguishing characteristic of which was that the inclination of the electrodes (which pierced the roof at an angle) could be varied at will in such a way that the furnace could be operated with radiating arc between the

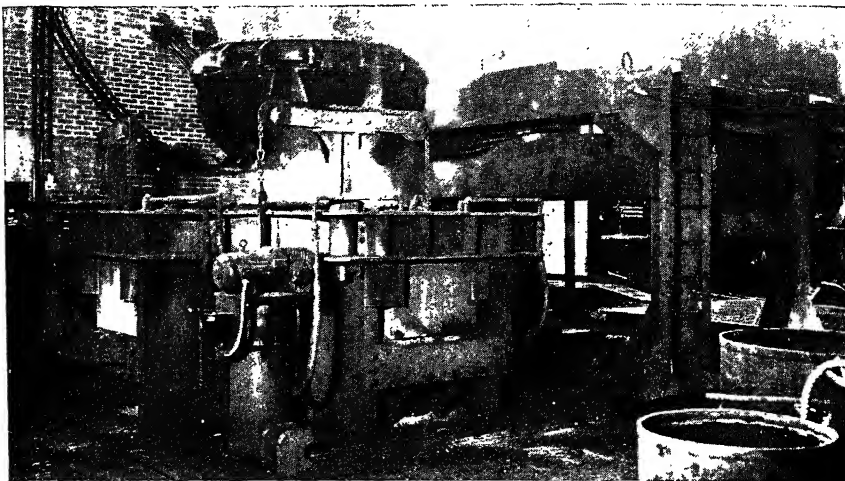


FIG. 24.—Modern Héroult-type furnace. (Courtesy of Samuel Arnold III, of Pittsburgh.)

two electrodes above the metal during the melting process, and then, when the bath was fluid, the electrodes could be tilted down and the arc struck between the electrodes and the bath. The furnace was of the tilting type. The working arc potential varied from 160 to 170 volts, which was considerably above the average of its period. This higher voltage, of course, makes possible smaller cross sections in electrodes

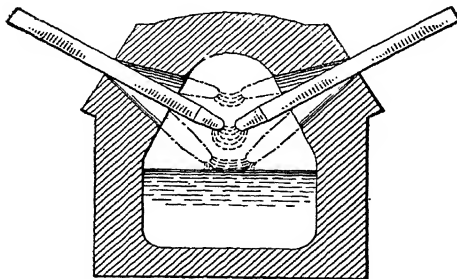


FIG. 25.—Bassanese arc furnace.

and bus bars with consequent smaller heat losses. It is equally true, however, that induction losses must be higher at these voltages. This furnace was chiefly employed in ferrous work, but was also adapted to nonferrous melting. A schematic vertical section is shown in Fig. 25.

Detroit Rocking Electric Furnace.—The Detroit indirect-arc rocking-type furnace¹ which approximates a central cylindrical section and two truncated cones in shape, is lined with insulation and refractory, the latter forming the hearth on which melting takes place. It employs two horizontal electrodes entering the furnace, one from either end, striking an arc between the electrodes in the center of the melting chamber. The cold or solid charge lies below and is "banked" front and rear of the electrodes with the initial melting by radiant heat from the arc. As the charge melts down, the angle of rock increases and the molten metal picks up heat from the lining which has been exposed to the direct radiation of the arc. The furnace on full rock rotates about 160 deg., keeping all the lining at metal temperature except the refractory adjacent to the charging door and about the electrode port sleeves. It is built in

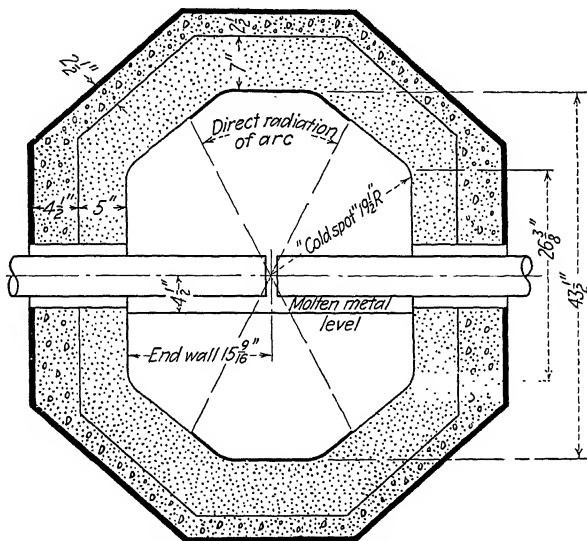


FIG. 26.—Detroit rocking furnace.

sizes from 6 to 4000 lb. cold charge or 8000 lb. molten bronze capacity. It is used in melting standard brass and bronze alloys, aluminum bronze, beryllium copper, nickel, zinc, alloyed and plain gray and white irons, special nonferrous alloys for cutting tools, carbon steel, low- and high-alloy and high-speed tool steels.

Exceptionally rapid melting is a feature of the rocking arc-type electric furnace.¹ The melting rate in a rocking arc furnace must be adjusted to the rocking action to ensure proper mixing action.

The "stagnant" atmosphere in the melting chamber is an advantage possessed by these units. It is not contaminated with products of combustion that might be harmful to the metal. The furnace atmosphere contains a considerable amount of carbon monoxide, which tends to preserve the composition as charged, due to the absence of oxidation. This feature can be a disadvantage in some cases where it is desired to oxidize impurities. The introduction of air to the chamber is not recom-

¹ ST. JOHN, April, 1944, meeting of the American Foundrymen's Association.



mended, and fluxes should not be used in the rocking-arc furnace unless absolutely necessary.

Any patches installed in the lining should be thoroughly dried to avoid the harmful effects resulting from moisture.

Silicon carbide refractories installed around the door opening are stated to be satisfactory, although with melting of the red brasses, it is said that difficulties may arise if the surface of the silicon carbide is not glazed.

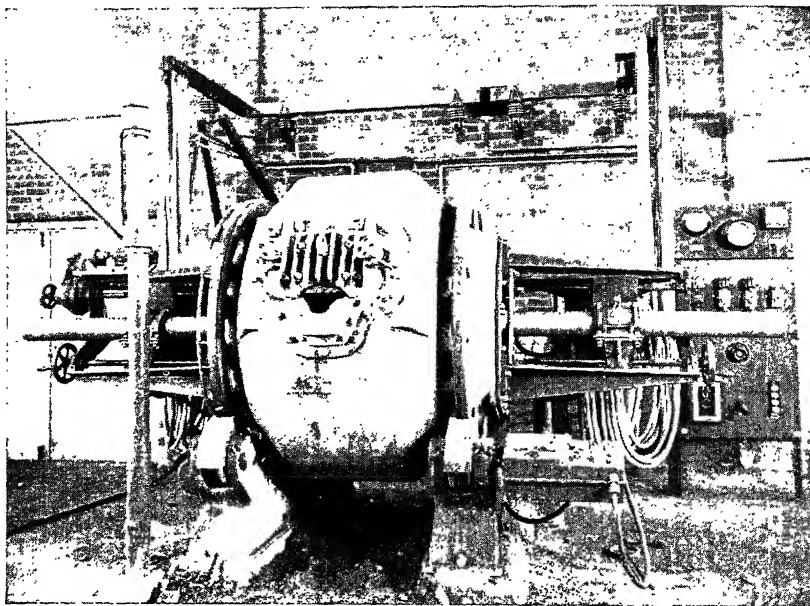


Fig. 27.—Detroit rocking furnace with mechanical electrode control

Power consumption is stated to be about 250–300 kw.-hr. per ton on continuous operation. Power consumption per ton is higher for intermittent melting. About 2.5 to 3 lb. of electrodes are consumed per ton of metal in this practice.

The Wile furnace¹ is of interest as it has been successfully used in smelting tin ores. It is of the shaft type and has three electrodes in the top and a heating element in the bottom. Like other shaft furnaces the Wile is nontilting and is continuous in operation. The furnaces are water-jacketed at the bottom, sides, and top.

The Krupp furnace² is a crucible-resistance tilting-type furnace. It consists essentially of a cylindrical shell with a refractory lining. Inside this lining is a concentric hollow cylinder of rammed Kryptol. Kryptol is the trade name given to a patented carbonaceous granular resistance material which, according to Fried. Krupp, is composed chiefly of broken-up electric-light carbons. Inside this hollow cylinder of rammed Kryptol is the crucible proper, usually made of rammed magnesite. The

¹ R. S. Wile, 92 Liberty St., New York, N.Y.

² FRIED. KRUPP A.G., Essen.

crucible is ordinarily not removable. The current is introduced by a carbon contact block below and a carbon contact ring above the Kryptol cylinder. The resistance of the Kryptol to the passage of the current heats the enclosed magnesite crucible

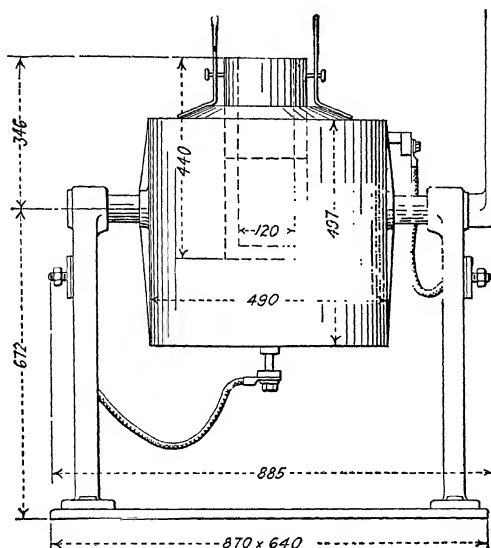


FIG. 28.—Krupp self-contained tilting furnace (dimensions in millimeters).

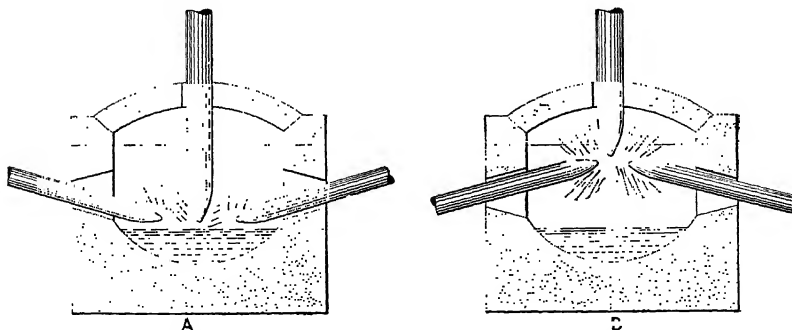


FIG. 29.—Rennerfelt furnace: A, direct-arc position, charge melted; B, free-burning position for melting.

and melts the charge. This furnace has the advantage over others of this type in that the contact blocks are more protected from oxidation and, further, that a greater economy is secured by keeping the crucible always in the furnace. On the other hand,

when it is necessary to renew a crucible, it is usually also necessary to renew partially the Kryptol resistor.

The Rennerfelt furnace¹ was very largely used in nonferrous work both in the United States and in Europe, but is believed to be used now only by the mints in the United States. It is of the indirect-arc type (sometimes called the free-burning arc) where the arc is not struck between the electrode and the bath, but between the electrodes. The furnace is also used on ferrous work. The indirect-arc type has much to recommend it in treating precious metals and metals of low melting point. Whenever a metal bath is allowed to come in direct contact with the carbon arc (*i.e.*, with no protecting slag between), there is sure to be a certain amount of volatilization and loss. This is largely avoided in the indirect-arc type. Figure 29 shows schematically

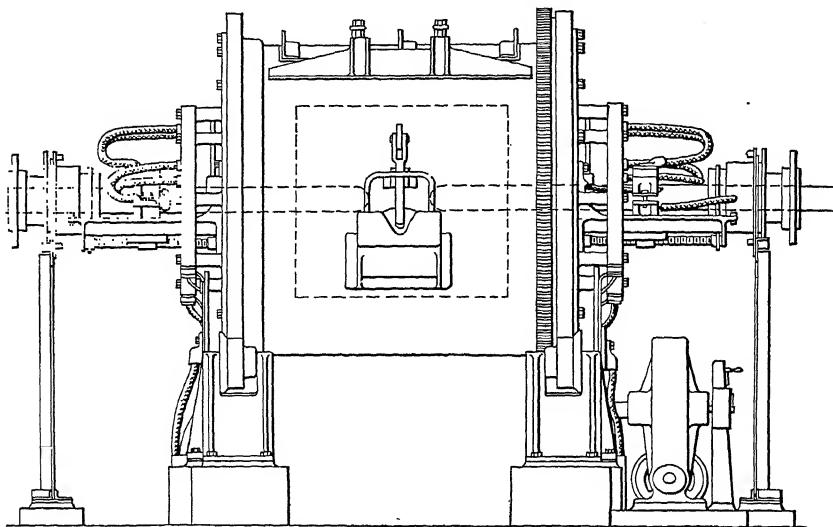


Fig. 30.—Weeks furnace.

the furnace in section. The furnace is provided with three electrodes. One of these pierces the roof, and the other two enter the furnace chamber from the sides. The arc is drawn from point to point of the three electrodes.

The Weeks furnace¹ (see Fig. 30) is of the rocking-barrel indirect-arc type. It consists essentially of a steel cylindrical shell, mounted on trunnions on its long axis. The trunnions are equipped with an automatic rocking mechanism for agitating the bath. As is usual in this type of furnace, the electrodes enter from the ends, and the arc is struck between the points. The connection is, of course, single phase. One door in the periphery of the furnace is used for charging, and the teeming door is located directly under it. The electrodes are water cooled and do not rotate with the furnace. The furnace rotates or "rocks" through an arc of 240 to 270 deg. The electrodes are automatically controlled.

The Moore "Lectromelt" furnace² is widely used throughout the world. It is of the direct-arc tilting type, and, while chiefly utilized in steel work, it is well

¹ General Furnace Co., Philadelphia, Pa.

² Pittsburgh Lectromelt Furnace Corp., Pittsburgh, Pa.

adapted to copper and nickel melting and refining. Three electrodes are employed, and three-phase connections used. The furnace is equipped with both hand and mechanical tilting mechanism and may be tilted forward for pouring and backward for slagging operations. Illustrations are given in Figs. 31 and 32. It is made in sizes to tap from 25 lb. to 100 tons per heat. The electrodes are automatically controlled, and, as with other furnaces of this type, the furnace may be governed by automatic current regulators to ensure an even temperature and power load. The furnace has a high power factor, ranging from 85 to 95 per cent. It is star connected on the secondaries with the neutral lead in contact with the bath. This arrangement tends to give a well-balanced load on the power line. The furnace is capable of high

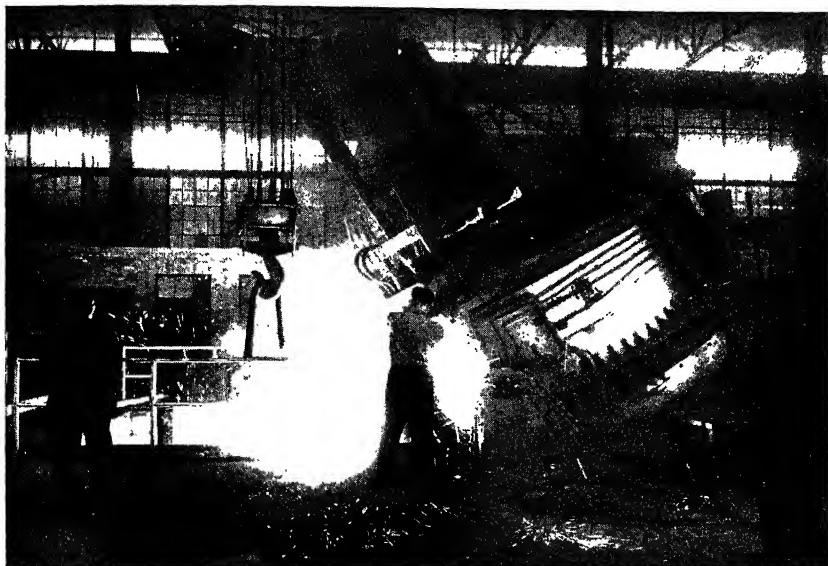


FIG. 31.—Lectromelt 20- to 25-ton furnace in tapping position.

energy efficiency. A 3-ton unit operated at the Huntington Works of the International Nickel Co. is reported by them to have averaged on intermittent operation 685.7 kw.-hr. per ton of refined Monel metal.

Advantage of Top-charge Furnaces.—The original top-charge furnaces had a tilting furnace top (the so-called coffeepot type), which threw a considerable strain on the roof of the refractory and led to a comparatively short life, so that for some time the top-charge furnace was in disfavor. With the present type of roofs rotating in a horizontal plane, this strain on the refractories is no longer a factor. Due to the shorter time between heats, a top-charge furnace has an increased capacity over the older type and does not use any more men, in fact sometimes a smaller furnace crew is employed. The manufacturers also claim a decreased electrode consumption per ton of output.

Other major improvements made in recent years to arc melting furnaces (in addition to electrode control already spoken of) are as follows:

The insertion of a water-cooling pipe of suitable diameter in the top row of brick of the side walls has given improvement to the life of the roofings and side-wall course of roof brick and the roofing (particularly in large furnaces). Use of roof rings of larger diameter (frequently larger than the furnace shell) has increased the life of roof rings and roof skew-back brick. A newly designed electrode cooling gland to be inserted *through the entire thickness* of the roof brick where the electrode enters through the furnace roof. This gland is thoroughly water-cooled and mechanically supported. Previously, cooling glands rested either directly on top of the brick or were inserted for only a very small portion of the depth of the brick. Thus the center section of the roof refractories between the electrodes was not thoroughly cooled, with resultant frequent failure of that part of the roof.

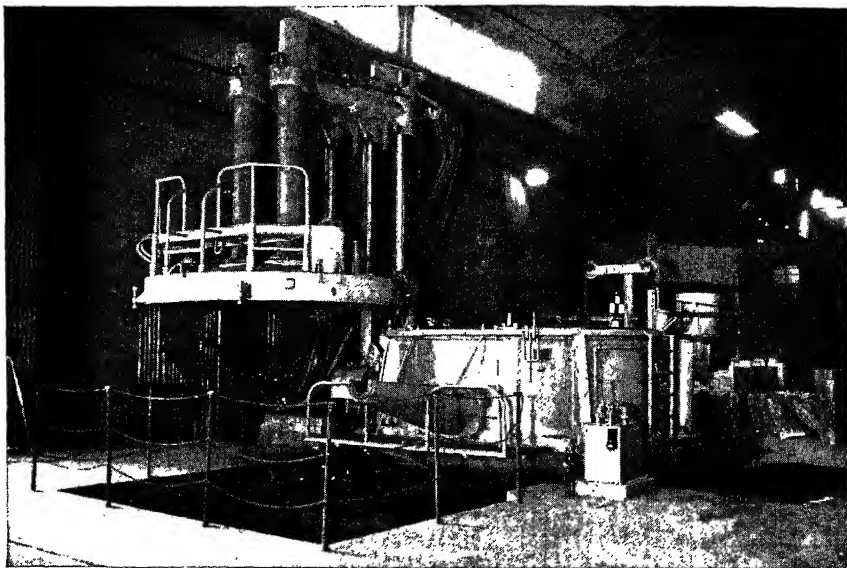


FIG. 32.—Lectromelt furnace with roof and superstructure swung aside for quick top charging.

Electric Heat-treating Furnaces.—Practically all furnaces of this type use some form of solid resistor. Arc furnaces of this type have been difficult to develop, owing to highly localized heating. Practically all such furnaces use a continuous solid or granular solid resistor. As a rule, the continuous solid resistors are limited to carbon, graphite, silicon carbide (Globar), and alloys of chromium and nickel, while the granular resistors are practically limited to various forms of carbon. Furnaces of this type differ little in principle, but greatly in design and operation.

Electric heat-treating furnaces may be roughly divided into low- and high-temperature types. Those from 900°F. down may be said to be low-temperature furnaces and are seldom used for metallurgical purposes. Therefore only the high-temperature types ranging upward from 900°F. will be considered here.

The various styles of furnace—batch or hearth, pusher, chain conveyer, car, rotary, etc.—developed in fuel-fired design are also to be found in electric design,

modified to suit the altered conditions brought about by the different method of introducing and developing the heat energy.

The outstanding characteristic of all these furnaces is a superiority of temperature control over the fuel-fired types. By means of thermocouples connected in circuit with the necessary controller, contactor switches, and relay, the current flow can be regulated to a high degree.

The resistors can be connected in many combinations, both parallel and series, delta and star.

In general, however, these furnaces do not pertain to the extraction of metals, and their treatment is out of place here.

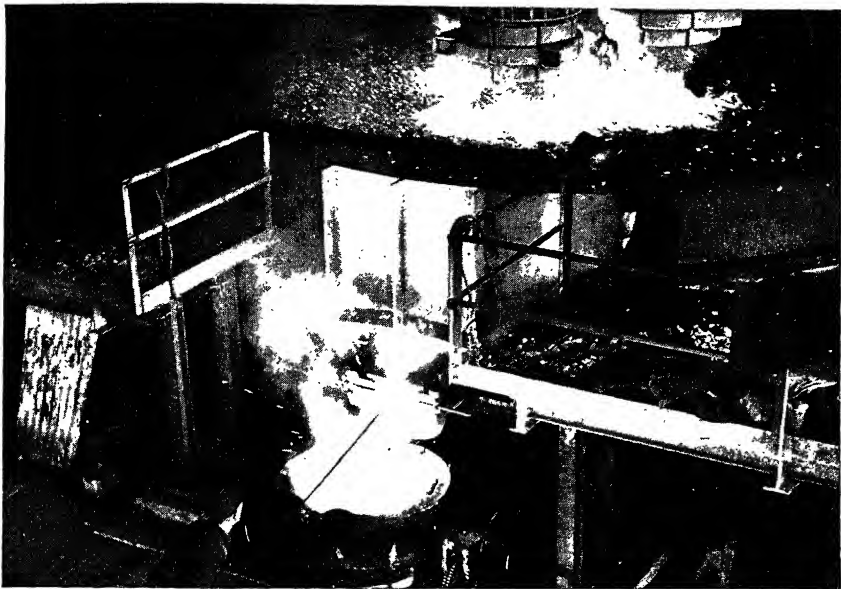


Fig. 33.—Lectromelt ferrosilicon furnace. (Courtesy of Pittsburgh Lectromelt Furnace Corp.)

Carbide and Silicide Furnaces.—Carbide and silicide furnaces are essentially reduction furnaces of the buried-arc resistor type. The chief representative of the carbides is calcium carbide, while the silicides are represented by carborundum, alundum, etc. These furnaces have several characteristics in common, the chief of which are the combination of buried-arc and resistance heating, large installations, high and fairly steady current demand, and long life of the electrode. Since calcium carbide becomes fluid or semifluid at furnace temperatures, the carbide furnaces have their electrodes enter from the top. On the other hand, the silicide furnaces are of the horizontal type, where the charge is built up around a core between two furnace "heads," through which project the horizontal electrodes. As calcium carbide and the various commercial silicides are controlled by patents or otherwise, and their manufacture is in the hands of a comparatively small number of companies, exact details of existing methods of construction and practice are in many cases not available for publication, although the general theory and procedure are well known. The

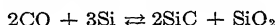
Carborundum Co. of Niagara Falls furnishes the following data on their furnace practice.¹

Furnace Construction.—The Carborundum furnace consists of two permanent brick heads containing water-cooled graphite electrodes by means of which current is introduced into the furnace. The sides of the furnace are built of firebrick usually held in cast-iron containers which can be removed rapidly in sections. A typical carborundum furnace using 2000 hp. has the following inside dimensions: length 40 ft., width 7½ ft., height 6½ ft. The graphite core is placed in the middle of the furnace between the two heads, this core having a cross section of approximately 3 sq. ft. The remainder of the furnace is filled with charge consisting of coke and sand.

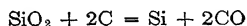
A typical charge is as follows: 15,000 lb. of coke, 25,000 lb. of sand, 3400 lb. of sawdust, and 700 lb. of salt. Sawdust is used to make the mass porous, to facilitate escape of gas, and the salt is used to aid the elimination of iron in the form of chloride. The principal reaction is



An additional reversed reaction probably takes place



Also



A typical furnace run lasts for approximately 36 hr. The average energy consumption during this period is 1500 kw. The voltage at the beginning of the run is 300 to 330 volts, and at the end of the run this generally drops to around 250 volts. These voltages are not fixed, as they vary with the resistance of the core. Electrode consumption in the manufacture of carborundum is not heavy. For instance, the carborundum furnace described above would have two 10-in. graphite electrodes in each head. These electrodes would be expected to last anywhere from 6 months to a year.

TABLE 1¹

Material	<i>p</i> at 15°C.	<i>a</i>
Aluminum.....	0.03–0.05	+0.0039
Lead.....	0.22	+0.0041
Iron.....	0.10–0.12	+0.0045
Copper.....	0.018–0.019	+0.0037
Brass.....	0.07–0.08	+0.0015
German silver.....	0.15–0.36	+0.0002–0.0004
Nickel.....	0.15	+0.0037
Platinum.....	0.12–0.16	+0.0024–0.0035
Silver.....	0.016–0.018	+0.0034–0.0040
Steel.....	0.10–0.25	+0.0052
Zinc.....	0.06	+0.0042
Carbon.....	100–1a000	–0.0003–0.0008

¹ Rodenhauser and Schoenawa.

The production of carborundum from a typical furnace run as described above varies considerably, but an average production of good abrasive grain would be 12,000 lb. per run. In addition to this there would be recovered from the furnace approximately 4000 lb. of second-quality carborundum and fire sand, which is used as a refractory material.

¹ O. Hutchins.

TABLE 2.—RESISTIVITIES AT HIGH TEMPERATURES¹

700°C. (932°F.)	Microhms, centimeter cubed	1000°C. (1832°F.)	Microhms, centimeter cubed	1500°C. (2732°F.)	Microhms, centimeter cubed
Silver, solid.....	5.0	Copper, solid.....	9.42	Silver, fused.....	23.0
Copper, solid.....	5.1	Gold, solid.....	12.54	Copper, fused.....	24.8
Gold, solid.....	6.62	Silver, fused.....	17.01	Aluminum, fused.....	29.0
Aluminum, solid.....	10.0	Aluminum, fused.....	24.0	Gold, fused.....	37.0
Brass, 2-1, solid.....	12.5	Molybdenum, solid.....	23.5	Molybdenum, solid.....	40.5
Molybdenum, solid.....	16.5	Tungsten (a), solid.....	30.5	Tungsten, solid.....	43.0
Tungsten (a), solid.....	18.0	Tungsten (b), solid.....	33.4	Tungsten (b), solid.....	50.0
Platinum (b), solid.....	25.3	Platinum (b), solid.....	40.8	Platinum (b), solid.....	52.6
Cadmium, fused.....	34.12	Brass, 2-1, fused.....	41.0	Tantalum, solid (b).....	74.4
Platinum (a), solid.....	34.4	Tantalum, solid.....	57.0	Tantalum, solid (a).....	78.0
Tantalum, solid.....	36.0	Platinum (a), solid.....	66.0	Tin, fused.....	80.5
Zinc, fused.....	36.60	Tin, fused.....	68.0	Platinum (a), solid.....	98.0
Iron (a), solid, about.....	52.0	Lead-tin alloy, fused.....	98.0	Iron (a), solid, about.....	131.0
Tin, fused.....	54.62	Ferronickel, solid.....	105.0	Calido, solid.....	136.0
Lead-tin alloy, fused.....	81.0	Iron (a), solid, about.....	111.0	Lead, fused.....	148.0
Ferronickel, solid.....	94.0	Calido, solid.....	122.0	Iron (b), fused.....	166.0
Lead, fused.....	102.85	Lead, fused.....	125.0	Ohms.....	Ohms
Calido, solid.....	109.0	Nichrome II.....	128.0	Graphite (b).....	0.00058
Krupp metal, solid.....	115.0	Antimony (b), fused.....	136.0	Graphite (a).....	0.00089
Nichrome II, solid.....	119.0	Bismuth, fused.....	167.5	Carbon (a).....	0.0016
Bismuth, fused.....	139.9	Graphite (b).....	Ohms	Carbon (a).....	0.0022
Antimony, solid.....	152.0	Graphite (a).....	0.00065	Carbon (b).....	0.0029
Graphite (b).....	0.00080	Carbon (d).....	0.00086	Nernst filament, about.....	0.5
Graphite (a).....	0.00084	Carbon (a).....	0.0021	Refrax.....	0.5
Carbon (a).....	0.0027	Carbon (c).....	0.0024	Silfrax B.....	0.7
Carbon (d).....	0.0028	Carbon (b).....	0.0030	Carbon grains (b).....	0.85
Carbon (c).....	0.0033	Carbon powder.....	0.12	Graphite grains.....	1.2
Carbon (b).....	0.0037	Silfrax B.....	0.84	Kryptol.....	3.4
				Alundum, about.....	750.0

¹ Carl Hering.

TABLE 2.—RESISTIVITIES AT HIGH TEMPERATURES.—(Continued)

500°C. (932°F.)	Microhms, centimeter cubed	1000°C. (1832°F.)	Microhms, centimeter cubed	1500°C. (2732°F.)	Microhms, centimeter cubed
Carbon powder.....	0.22	Sodium chloride, fused.....	0.90		
Silicon.....	0.004 to 0.23	Glass, roughly about.....	1.0		
Lead chloride, fused, 520°.....	0.23	Graphite grains.....	1.7		
Silver chloride, fused.....	0.418	Carbon grains (b).....	1.9		
Lead chloride, solid.....	0.547	Carbon grains (a).....	2.8		
Silfrax B.....	0.824	Silicon powder.....	3.5		
Copper chloride, fused.....	0.92	Refrax.....	3.7		
Graphite grains.....	2.50	Kryptol.....	4.8		
Carbon grains (b), about.....	2.70	Porcelain, about.....	15.0		
Carbon grains (a), about.....	4.8	Manganese oxide powder.....	15.7		
Kryptol.....	8.5	Copper oxide (CuO), powder.....	18.0		
Refrax.....	10.0	Zinc oxide powder.....	26.7		
Boron, about.....	19.7	Iron oxide (Fe ₂ O ₃), powder.....	31.4		
Silicon powder.....	60.0	Quartz.....	110.0		
Glass, about.....	120.0	Magnesium oxide powder.....	1,400.0		
Iron oxide (Fe ₂ O ₃), powder.....	330.0	Alundum.....	8,000.0		
Copper oxide (CuO), powder.....	1,260.0				
Manganese oxide (MnO ₂), powder.....	1,570.0				
Copper oxide (CuO).....	2,200.0				
	5,640.0				

TABLE 3.—RESISTIVITY OF METALS
(Microhms per cubic centimeter)

	-160°	0°	18°	100°	Temperature coefficient at 0°
Aluminum.....	0.81 ^a	2.8	2.94	4.13	0.0040
Antimony.....	36.0	40.5	0.0041
Bismuth.....	55.55	119.0	160.3	0.0035
Cadmium (drawn).....	2.72	7.0	7.54	9.82	0.0042 ^a
Copper (drawn).....	0.49	1.58	1.78	2.36	0.0039
Calcium.....	7.5	10.5
Cobalt.....	9.71	0.0033 ^a
Gold.....	0.68	2.42	3.11	0.0037
Arsenic.....	33.3
Iridium.....	5.3
Iron.....	9-15	16.8	0.0062
Iron (wrought).....	5.4	13.9	18.8	0.0058
Lead (drawn).....	7.43	19.0	20.8	27.7	0.0039
Lithium.....	8.4
Magnesium.....	4.35	0.0038
Mercury.....	94.07	95.57	0.00072
Molybdenum.....	4.1 ^b	0.0050 ^a
Steel.....	19.9	25.6
Nickel.....	5.9	11.8	15.7	0.0062 ^a
Osmium.....	9.5
Palladium.....	10.7	13.8	0.0035 ^a
Platinum.....	2.4 ^d	9.0	11.0	14.0	0.0037 ^a
Potassium.....	6.64
Rhodium.....	6.0
Silver.....	0.56	1.50	1.65	2.13	0.00377
Sodium.....	4.74
Strontium.....	25.0 ^c
Tantalum.....	14.6	0.0033 ^a
Tellurium.....	21.0 ^c	0.0040
Thallium.....	17.6
Thorium.....	40.1
Tin (drawn).....	3.5	10.0	11.3	15.3	0.0043
Tungsten (annealed).....	4.42	4.81	6.65	0.0051 ^a
Zinc.....	2.2	5.6	6.1	7.9	0.00365

^a At -183°. ^b At 25°. ^c At 20°. ^d At -204°. ^e From 18 to 100°.

The values at low temperatures are mostly Lee's; those at 18°, Jaeger and Disselhorst's; those at 0° from a table compiled by Watt, "Laboratory Course in Electrochemistry," while those at 100° are from various sources.

TABLE 4.—RELATIVE CONDUCTIVITIES OF METALS FOR HEAT AND ELECTRICITY¹

Metal (in vacuo)	Heat	Elec- tricity	Metal (in vacuo)	Heat	Elec- tricity
Silver.....	100	100	Iron.....	11.9	14.44
Copper.....	74	77.43	Steel.....	10.3	
Gold.....	54.8	55.19	Platinum.....	9.4	10.53
Aluminum.....	31.33	Lead.....	7.9	7.77
Zinc.....	28.1	27.39	German silver.....	6.3	6.0
Brass.....	24	22.0	Antimony.....	4.03	
Cadmium.....	20.06	Bismuth.....	1.8	1.8
Tin.....	15.4	11.45	Mercury.....	1.3	

¹ LIPDELL, "Metallurgists' and Chemists' Handbook."TABLE 5.—RELATION OF HEAT AND ELECTRIC CONDUCTIVITY¹

Material	Thermal conductivity	Temperature coefficient of this ratio, per cent
	Electrical conductivity at 18°C.	
Copper, commercial.....	6.76×10^{10}	
Copper (1), pure.....	6.65×10^{10}	0.39
Copper (2), pure.....	6.71×10^{10}	0.39
Silver, pure.....	6.86×10^{10}	0.37
Gold (1), pure.....	7.27×10^{10}	0.36
Gold (2), pure.....	7.09×10^{10}	0.37
Nickel.....	6.99×10^{10}	0.39
Zinc (1).....	7.05×10^{10}	0.38
Zinc (2), pure.....	6.72×10^{10}	0.38
Cadmium, pure.....	7.06×10^{10}	0.37
Lead, pure.....	7.15×10^{10}	0.40
Tin, pure.....	7.35×10^{10}	0.34
Aluminum.....	6.36×10^{10}	0.43
Platinum (1).....	7.76×10^{10}	
Platinum (2), pure.....	7.53×10^{10}	0.46
Palladium.....	7.54×10^{10}	0.46
Iron (1).....	8.02×10^{10}	0.43
Iron (2).....	8.03×10^{10}	0.44
Steel.....	9.03×10^{10}	0.35
Bismuth.....	9.64×10^{10}	0.15
Constantan (60 Cu, 40 Ni).....	11.06×10^{10}	0.23
Manganin (84 Cu, 4 Ni, 12 Mn)...	9.14×10^{10}	0.27

¹ Jager and Diesselhorst.

TABLE 6.—CONVERSION TABLE OF UNITS OFTEN USED IN ELECTRIC-FURNACE CALCULATIONS¹

1 kw.-hr. =	{	1,000 watt-hr.
		1.34 hp.-hr.
		2,654,200 ft.-lb.
		3,600,000 joules.
		3,412 B.t.u.
		860 cal.
		367,076 kg.-m.
1 hp.-hr. =	{	0.235 lb. carbon oxidized with perfect efficiency.
		3.51 lb. water evaporated from and at 212°F.
		22.75 lb. of water raised from 62 to 212°F.
		0.746 kw.-hr.
		1,980,000 ft.-lb.
		2,545 heat units.
		273,834 kg.-m.
1 lb. carbon oxidized with perfect effi- ciency =	{	0.175 lb. carbon oxidized with perfect efficiency.
		2,618 lb. water evaporated from and at 212°F.
		17.0 lb. water raised from 62 to 212°F.
		14.544 heat units.
		1.11 lb. anthracite coal oxidized.
		2.5 lb. dry wood oxidized.
		21 cu. ft. illuminating gas.
1 lb. water evapo- rated from and at 212°F. =	{	4.26 kw.-hr.
		5.71 hp.-hr.
		11,315,000 ft.-lb.
		15 lb. of water evaporated from and at 212°F.
		0.285 kw.-hr.
		0.382 hp.-hr.
		971.7 heat units.
	{	104,580 kg.-m.
		1,026,000 joules.
		756,182 ft.-lb.
	{	0.06695 of carbon oxidized.

¹ National Carbon Co.

TABLE 7.—PHYSICAL PROPERTIES OF CARBON¹

Chemical symbol = C

Atomic weight = 12.010

Volatilization point = 3925–3970°K., 3652–3697°C., 6606–6687°F.

Melting point: Carbon, in all forms, volatilizes without melting at ordinary pressures.

Specific Gravity of Carbon Materials Used in the Production of Carbon Products

Gas-calcined petroleum coke carbon.....	1.98–2.10*
Gas-calcined lampblack carbon.....	1.80–1.85*
Gas-calcined anthracite.....	1.79*
Electrically calcined anthracite.....	1.90–1.97*
Graphitized petroleum coke carbon.....	2.20–2.24*
Graphitized lampblack carbon.....	1.98–2.08*
Graphite (pure).....	2.25
Diamond.....	3.51

Electrical Resistivity of Formed Carbon or Graphite Products at 20°C.

	Ohms per cubic inch†	Ohms per cubic centimeter†
Carbon, petroleum coke base.....	0.0014–0.0018	0.0035–0.0046
Graphite, petroleum coke base.....	0.0003–0.0005	0.0008–0.0013
Carbon, anthracite base.....	0.0013–0.0026	0.0033–0.0066
Carbon, lampblack base, gas baked.....	0.0023–0.0032	0.0058–0.0081
Carbon, lampblack base, graphitized.....	0.0018–0.0026	0.0046–0.0066

Specific heat

	Temperature, degrees centigrade					
	26–76	26–282	26–538	36–902	47–1193	56–1450
Carbon, anthracite, or petroleum coke base.....	0.168	0.200	0.234	0.315	0.352	0.387
Graphite, petroleum coke base...	0.165	0.195	0.234	0.324	0.350	0.390
Diamond.....	0.160	0.315	0.415			

¹ National Carbon Co.

* Dependent on source and degree of calcination.

† This property is also expressed as ohm-inches or ohm-centimeters.

TABLE 8.—STANDARD AMORPHOUS CARBON ELECTRODE DATA¹
Round Electrodes

Diameter, inches	Area, square inches	Current-carrying capacity, amperes ²	Current density, amperes per square inch
8	50	2,000–3,000	40–60
10	79	3,000–4,800	40–60
12	113	4,500–6,800	40–60
14	154	5,400–8,500	35–55
17	227	7,900–12,500	35–55
20	314	11,000–17,300	35–55
24	452	15,000–25,000	35–55
30	707	24,700–35,300	35–50
35	962	28,800–38,400	30–40
40	1,257	37,700–50,200	30–40

Square Electrodes

8	64	2,500–3,800	40–60
10	100	4,000–6,000	40–60
12	144	5,500–8,500	40–60
14	196	6,800–10,800	35–55
16	256	9,000–14,000	35–55
20	400	14,000–22,000	35–55
24	576	20,200–25,900	35–45
24 and 30	720	21,600–28,800	30–40

¹ National Carbon Co.

² On account of the varying conditions under which electrodes are used—whether open arc or submerged arc, whether duplex system or melting down from cold scrap—and on account of the many local variations in similar practices, it is impossible to do more than suggest general limits for current-carrying capacities.

TABLE 9.—POWER CONSUMPTION OF FERROALLOY FURNACES

Alloy	Grade product, per cent	Size of furnace		No. of phases	Electrode, volts	Per cent recovery	Kilowatt-hours per pound alloy tapped
		Tons, charged	Transfer, kilowatts				
Ferrouanium.....	35–50 U	3½	75	1	35–65	75	3–5
Ferrotungsten.....	70–75 W	¾	150	1	95	80–90	Smelting 2.5–3.0 Refining 2.0–3.5 Total 4.5–6.5
Ferromolybdenum....	60–65 Mo	1	150	1	65	78–80	4–5
	1.5–2 C						
Ferrovandadium.....	30–35 V	1	150	2	65	75 average	3.4
	3–4 Si						
Ferrosilicon.....	50–70 Si	3–4	1,000	1	60–90	2.75–5
Ferrocchrome.....	60–65 Cr	13–15	750	3	120	70–80	3–3.5
Ferromanganese.....	75–80 Mn	15	1,200	3	72	70–85	1.4–1.5

TABLE 10.—STANDARD SIZES CARBON NIPPLES¹

Nominal size of nipple, inches	For electrode diameter, inches	Top of thread diameter, inches	Length of nipple, inches	Depth of socket, inches
4 × 8	8	4 $\frac{3}{8}$	9 $\frac{1}{8}$	4 $\frac{3}{4}$
5 × 10	10	5 $\frac{3}{8}$	11 $\frac{1}{8}$	5 $\frac{3}{4}$
6 × 12	12	6 $\frac{3}{8}$	13 $\frac{1}{8}$	6 $\frac{3}{4}$
7 × 14	14	7 $\frac{1}{2}$	14	7 $\frac{3}{16}$
8 $\frac{1}{2}$ × 17	17	9 $\frac{1}{16}$	17	8 $\frac{1}{16}$
10 × 20	20	10 $\frac{1}{16}$	20	10 $\frac{1}{16}$
10 $\frac{7}{8}$ × 24	24	11 $\frac{5}{8}$	24	12 $\frac{3}{16}$
15.32 × 30	30	16.32	30	15 $\frac{3}{16}$
18 $\frac{1}{2}$ × 35	35	19.50	35	17 $\frac{1}{16}$
21 × 40	40	21.75	40	20 $\frac{3}{16}$

¹ National Carbon Co.

TABLE 11.—STANDARD ACHESON GRAPHITE ELECTRODES

Diameter, inches	Area, square inches	Current-carrying capacity, amperes	Current density, amperes per square inch
2	3.14	600–1,000	200–320
2 $\frac{1}{2}$	4.91	800–1,500	160–310
3	7.07	1,200–2,100	170–300
4	12.6	1,800–3,000	140–240
5 $\frac{1}{8}$	20.6	2,300–4,100	110–200
6	28.3	3,100–5,400	110–190
7	38.5	4,200–6,900	110–180
8	50.3	5,500–9,000	110–180
9	63.6	6,400–10,800	100–170
10	78.5	7,800–12,500	100–160
12	113	11,300–17,000	100–150
14	154	15,400–21,500	100–140
16	201	20,100–26,100	100–130
17	227	22,700–28,400	100–125
18	254	25,500–30,500	100–120
20	314	28,300–34,600	90–110
24	452	32,500–40,500	70–90
30	707	38,000–60,000	55–85

Square Electrodes

2	4	650–1,000	160–250
4	16	1,900–3,000	120–180
6	36	3,600–4,900	100–140
8	64	6,400–8,300	100–130
10	100	9,000–1,300	90–130
12	144	13,000–17,300	90–120
16.35	267	21,400–26,700	80–100
17.35	301	22,500–27,000	75–90

CHAPTER XVIII

APPLIED ELECTROCHEMISTRY

BY COLIN G. FINK, PH.D.¹

Introduction.—Industrial electrochemistry is based on the phenomena of electrolysis, electrothermics, and electronics. As to electrolysis, when a direct current is passed through a chemical compound either in solution or fused, if the voltage exceeds a certain minimum, decomposition will take place and the composing elements will be set free at the two electrodes or new combinations will be formed with or at the electrodes.

In electrothermics, the high temperatures of the electric-arc or electric-resistor furnace are used in bringing about reactions and forming products impossible or far more costly by fuel-fired furnace methods.

That phase of the electrochemical industry based on electronics is largely concerned with the electric discharge through gases—the production of ozone, the Cottrell precipitator, etc.

Electrochemical processes have displaced older chemical or metallurgical processes on various accounts: electrochemical processes will turn out pure products at lower cost, *e.g.*, copper and aluminum; and will turn out products such as silicon carbide or silicon metal never before possible on a commercial scale. Furthermore, in many cases electrochemical processes will utilize hydroelectric power direct with the complete elimination of coal or other fuel such as used in older processes. After all, coal and oil are becoming too valuable as raw materials for the chemical industry to be wasted as fuel. In this connection, the nationwide Federal Irrigation Projects such as those of Bonneville, Grand Coulee, and Boulder have brought the cost of electric energy down to a few mills per kilowatt-hour, greatly stimulating the substitution of hydroelectric power for fuel-electric power.

The electrochemical industries today comprise the following:

1. The electrowinning of copper, zinc, cadmium, and manganese from leach liquors.
2. The electrolytic refining of copper, lead, tin, bismuth, silver, and gold.
3. The electrolytic decomposition of sodium chloride into sodium and chlorine and of water into hydrogen and oxygen.
4. The electrolytic reduction or oxidation of compounds, such as the reduction of glycose to mannitol; and the oxidation of chromous to chromic salts.
5. The electroplating of copper, silver, gold, nickel, cobalt, chromium, rhodium, palladium, cadmium, zinc, iron, tin, lead, as well as of alloys such as brass on various basis metals such as steel, copper, brass, and other metals.
6. The production of aluminum, magnesium, sodium, and other metals by the electrolysis of fused-salt baths.
7. The electric-furnace production of metals such as magnesium and zinc, of alloys such as steel and ferrosilicon, of graphite, carbides, cyanamide, phosphorus, carbon bisulphide, and abrasives.
8. The production of primary cells and storage batteries—one of the largest of the electrochemical industries.
9. The electric discharge through gases, as in the Cottrell precipitator or in the production of ozone or of fluorescent light.

¹ Columbia University, New York.

Electrochemical methods applied to nonferrous metallurgy have greatly increased in scope and importance. Of 25 of the metals covered in "Recovery of the Metals," 18 are produced wholly or partly by electrochemical methods. Three of these nonferrous metals, aluminum, magnesium, and sodium, are now next to iron the cheapest available commercial metals (on the unit-volume basis).

The electrometallurgical methods applied may be divided into three groups:

1. The electrolysis of aqueous solutions of metal salts.
2. The electrolysis of molten or fused salts.
3. Smelting in the electric furnace.

This chapter is confined to (1) and (2). Topic (3) is covered in Chap. XVII.

No matter what metal is being electrochemically refined or won, whether copper or nickel, or aluminum or magnesium, in every case it is of prime commercial importance to refine or recover such metal at a minimum cost and at the same time maintain the purity and high quality of the metal specified.

The cost of producing any metal electrochemically is the sum total of a number of items, such as labor, rent, and interest on investment. The cost item we are going to deal with in particular is the electric power cost of producing, say, a ton of copper or a ton of zinc. The electric power unit is the kilowatt-hour which is the product of amperes times volts times hours.

Electrical Units.—Neglecting the c.g.s. definitions, the practical electrical units may be defined as follows: The *ampere* is the current that when passed through a silver nitrate solution in water will deposit 0.001118 g. of silver per second. One ampere passing for 1 sec. is defined as a *coulomb*. The *ohm* is the unit of resistance. It is the resistance offered by a column of mercury at 0°C., 14.4521 g. in mass, of a constant cross section and 106.3 cm. in length (also defined as the resistance of a column 106.29 cm. long, 1 sq. mm. in cross section at 0°C.). The *volt* is the unit of pressure; 1 volt will produce a steady current of 1 amp. through 1 ohm of resistance. The *watt* is the unit of energy; 1 amp. flowing for 1 sec. under a pressure of 1 volt requires the expenditure of 1 watt of energy.

The Electrolytic Cell.—In its simplest form an electrochemical cell consists of a container for the electrolyte (usually an aqueous solution or a fused salt) and two electrodes, *viz.*, a cathode and an anode. The cathode is that pole or electrode

1. That emits electrons.
2. To which positive charged ions (cations) migrate under the influence of the electric current.
3. At which metals are deposited.
4. At which hydrogen is evolved.
5. At which reduction takes place.
6. At which negative charged hydroxyl ions (OH^-) are formed.
7. That has an alkaline surface film.

Conversely the anode is that pole or electrode

1. That receives or accepts electrons.
2. To which negative charged ions (anions) migrate under the influence of the electric current.
3. At which metals go into solution.
4. At which oxygen and chlorine are evolved.
5. At which oxidation takes place.
6. At which positive charged hydrogen ions (H^+) are formed.
7. That has an acid surface film.

An ion of sodium, Na^+ , is an atom of sodium that has acquired a positive charge by losing an electron. An ion of chlorine, Cl^- , is an atom of chlorine that has taken on an additional electron and acquired a negative charge.

As to the amount of current required to electrodeposit a given weight of metal, this can never be less than a certain minimum number of coulombs, this minimum depending on the given metal's weight and the metal's valence. This exact relation between the weight of metal deposited and the current consumed was discovered by Faraday over 100 years ago and forms the basis of our power-cost determination per pound or ton of metal produced.

Electrochemical Equivalents.—According to Faraday, 96,500 coulombs (equivalent to 26.8 amp.-hr.) are required to electrodeposit 31.78 g. of copper from a cupric-salt solution. Therefore 1000 amp.-hr. will deposit 2.614 lb. of copper. The amount of any given substance that is set free, provided a certain minimum voltage is exceeded, is a function only of the time and the amperes of current that are passed through the electrolyte, and this quantity is not altered by the combination in which the given substance exists, provided the valence is not altered. Thus, 0.01976 g. of copper is deposited when 1 amp. passes through a cupric salt for 1 min., whether it is cupric sulphate, cupric nitrate, or cupric chloride. In the same way, 0.06709 g. of silver will be deposited by 1 amp. flowing for 1 min. and 60×0.06709 g., or 4.0258 g., by 1 amp. flowing for 1 hr. These electrochemical equivalents are proportional to the atomic weights divided by the valence. They are given on pages 644 and 645 of the Appendix.

Current Efficiency.—The values tabulated represent the maximum weight of metal deposited on the cathode under ideal conditions of solution composition, temperature, circulation, etc. In actual practice, because of the fact that the commercial solutions of copper, nickel, zinc, etc., contain many other salts besides that of the particular metal we wish to recover and that there are other disturbing factors, the actual weight of any metal electrodeposited per ampere-hour, or per 1000 amp.-hr., is always less than the theoretical amount indicated by Faraday's law, shown in Table 1. Thus in the case of copper refining the actual weight of copper deposited on the cathodes is usually not more than 96 per cent of the weight indicated in the table. This value of 96 per cent is termed the "current efficiency." Ordinarily this current efficiency is determined by noting the total ampere-hours consumed in the deposition of a weighed amount of copper, or nickel, or whatever other metal is being electro-deposited, and dividing this number of actual ampere-hours into the number of ampere-hours theoretically required (Appendix). In the laboratory, for more exact determination of the current efficiency the experimental metal deposition cell is connected in series with a cell termed a "coulometer" which latter is nothing other than a metal deposition cell operating under such ideal conditions that the deposition of metal at the cathode is in strict accordance with the laws of Faraday.

The coulometer that has given the most accurate results is the silver coulometer. The internationally accepted value for the ampere is based on this silver coulometer. "The International ampere is defined as the current which will deposit silver at the rate of 0.00111800 g. per sec." For general test purposes the copper coulometer is sufficiently accurate. It comprises a pure copper-sheet cathode of any convenient size, suspended between two pure-copper anodes, slightly smaller in surface area than the cathode. The electrolyte is composed of $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$, 150 g. per l.; H_2SO_4 , 50 g. per l.; $\text{C}_2\text{H}_5\text{OH}$, * 50 g. per l.

Current Density.—It will be readily appreciated that since there exists, according to Faraday, a relation between current consumed and metal deposited at the cathode

* Without the addition of the ethyl alcohol, the copper deposit is always appreciably less than Faraday's law requires. This is attributed to the interaction of cuprous ions, formed during electrolysis, and the dissolved oxygen. In other words, the electrodeposition process $\text{Cu}^{++} \rightarrow \text{Cu}^+ \rightarrow \text{Cu}^0$ (cupric ions to cuprous ions to copper metal) is interfered with by the dissolved oxygen which reoxidizes the cuprous to cupric $\text{Cu}^+ \rightarrow \text{Cu}^{++}$ and thereby lowers the current efficiency. The addition of alcohol eliminates this error because it has a greater avidity for the dissolved oxygen than has the cuprous ion,

the general overhead charges included in the cost of producing the electrolytic metal will be decreased by increasing the current through the cells and, consequently, the output of metal per cell per unit time. In other words, by doubling the current we should expect twice the output per cell.

The current passing through the cells is customarily expressed in amperes per unit area (square foot) of cathode surface; this expression is termed "current density." In practice it has been found that for most of the common metals such as copper, nickel, and zinc there is an upper limit to the current density which if exceeded will adversely affect the quality of the cathode deposit or the power cost per unit weight of metal produced. This upper limit is usually around 40 to 50 amp. per sq. ft. for stationary electrodes. In normal times the copper refineries and other electrolytic metal plants operate at current densities between 15 and 25 amp. per sq. ft.

If excessive current densities are used, a serious drop in current efficiency results. And furthermore, because of interfering reactions at the cathode, impure metal of inferior physical properties is produced.

In general, the best current density to apply in any metal electrodeposition process, such as electrefining or electrowinning, will depend on

1. Composition and concentration of the electrolyte.
2. Effective circulation.
3. Temperature of the electrolyte.
4. Free acid content of the electrolyte.
5. The market price of the metal produced.

Electrolyte composition in large-scale practice is almost always complex. In rare instances, such as the electrolytic recovery of cadmium as a by-product in the electrolytic zinc industry, are the constituents of the electrolyte merely two or three in number. In most cases this is not true, *e.g.*, in the electrolytic refining of copper, the electrolyte contains the following metals and compounds: Cu, Ni, Co, Fe, Sn, Zn, Na, K, Ca, Mg, As_2O_3 , Bi_2O_3 , SbO, besides traces of Se, Te, Au, Pt, Pd, and Ag.

Of all the cell-solution constituents in this case, we are interested primarily in the copper, but it is absolutely necessary in plant practice to take into consideration the impurities present, in particular nickel and cobalt, arsenic and silver in the case of copper refining. For example, to prevent silver from codepositing with the copper, a little common salt, NaCl, is added to the electrolyte to hold back the silver at the anode as insoluble AgCl. Nickel, cobalt, and arsenic are removed from the solution by crystallization and fractionation. The concentration of the electrolyte is very nearly that of saturation so that an excessive temperature drop of the electrolyte of, say, 20°C. will cause sulphate of copper and other salts to crystallize out in the launders and pipes, causing serious interference with the pumping and circulation.

By "effective circulation" we imply that the circulation of the electrolyte is sufficiently rapid and voluminous to counteract, as far as practical, the depletion of metal ions next to the cathode and counteract consequent concentration polarization. It has been found by many years of experience in copper, in nickel, in zinc, and in other electrolytic metal plants that it is advisable to circulate the electrolyte at the rate of about 16 l. per min. Higher rates are prohibitive in an electrolytic refining of copper and other metals on account of the stirring up of the anode slimes which will deposit on the cathode, resulting in low-grade copper.¹

Temperature of the Electrolyte.—As to the relation between current-carrying capacity and temperature of the electrolyte, it has been found in actual practice that for a given voltage between terminals of an electrolytic cell, such as a copper-refining

¹ The slimes comprise fine particles of insoluble compounds such as cuprous sulphide or cuprous selenide and arsenic compounds, which go into suspension as a result of excessive circulation and thereupon migrate to the cathode (by cataphoresis) and codeposit with the copper, resulting in metal of low electrical conductivity.

cell, the amperes that will pass through the cell increase as the temperature of the electrolyte is increased. In other words, a hot metal-sulphate solution is a better electrical conductor than a cold solution. Accordingly, all electrorefining cells are kept at an elevated temperature, usually 55 to 65°C. It might be asked, "Why not at 90 to 100°?" The answer is that what is then gained in lower ohmic resistance of the bath is more than offset by solution evaporation losses and excessive contamination of the cell-room atmosphere, added to the extra cost of heating the electrolyte to the higher temperatures.

Ion Mobilities.—The reason for the better current-carrying capacity of a metal-salt solution at elevated temperature is due to increased speed or mobility of the ions present in the electrolyte, even though these ions are fewer in number at high temperature than at low temperature.

We stated above that the fourth factor on which proper selection of current density is based is "free-acid content of the electrolyte." It is customary in electrolytic metal refining to make the electrolyte as stable as possible and also as good a conductor as possible. It will be remembered that the solution film directly in contact with the cathode surface is decidedly alkaline as can easily be demonstrated with a drop or two of phenolphthalein solution. This alkaline film will frequently give rise to the precipitation of basic-metal compounds which are relatively insoluble and which will codeposit with the metal being refined. To offset this basic-salt formation and, furthermore, to increase the conductivity of the electrolyte, acid, such as sulphuric acid, is added to the bath. The hydrogen ions thus introduced conduct the current over seven times as fast as copper ions, nickel ions, or zinc ions do.

TABLE 1.—ION MOBILITIES AT 25°C.

	Centimeters per Second
Hydrogen H^+	36.2×10^{-4}
Sodium Na^+	5.19×10^{-4}
$\frac{1}{2}$ Copper Cu^{++}	4.9×10^{-4}
$\frac{1}{2}$ Nickel Ni^{++}	4.9×10^{-4}
$\frac{1}{2}$ Zinc Zn^{++}	4.8×10^{-4}
Hydroxyl OH^-	20.5×10^{-4}
Sulfate SO_4^{--}	8.27×10^{-4}
Chloride Cl^-	7.91×10^{-4}

We have given the ion mobilities of eight of the ions usually met with in electro-metallurgy. It will be noted that hydrogen and hydroxyl ions are the fastest moving ones. Accordingly, the hydrogen ions present in the bath serve a double purpose: (1) They reduce the probabilities of forming basic salts at the cathode surface, and (2) they increase the conductivity of the electrolyte, thereby reducing the electric power cost. The last factor cited as determining, or affecting, the current density is the market price of the metal produced. Briefly, the higher the demand and the higher the market price of the metal, the higher is the current density applied. On the other hand, it will be appreciated that the entire process, from ore to finished metal, must be kept in step. Little is gained by doubling the output of the electrolyte cells (by doubling the current density) and then finding that the supply of "raw" copper such as anode copper or blister copper is exhausted after a short time of operation of the cells at double current density.

Cell Voltage.—We have pointed out above that there is an exact relation between current input of an electrodeposition cell and metal deposited at the cathode (laws of Faraday). No such close relation exists between the metal deposited and the voltage of the cell. But there is this much, *viz.*, that a minimum cathode potential is required

to discharge and/or deposit any cation in solution and this minimum potential varies from metal to metal as well as from solution to solution.

The decomposition potential of copper sulfate, for example, in aqueous solution is that minimum potential required under ideal conditions to deposit copper continuously at the cathode and discharge sulphate ions at the anode.

TABLE 2.—DECOMPOSITION POTENTIALS¹

Normal Aqueous Solution of	Decomposition Voltage, Volts
Zinc sulphate.....	2.55
Nickel sulphate.....	2.09
Cadmium sulphate.....	2.03
Cobalt sulphate.....	1.92
Copper sulphate.....	1.49
Silver sulphate.....	0.80
Sulphuric acid.....	1.67

¹ CREIGHTON, H. J., "Principles of Electrochemistry," John Wiley & Sons, Inc., New York, 1935.

The decomposition potentials recorded in the table are specifically for 1 normal sulfate solutions. The order of the metal is in increasing "nobility"; *i.e.*, it takes over 2.55 volts to decompose a zinc sulphate solution and less than one-third of this potential to decompose a silver sulphate solution. However, the order of sequence for the metals in 1*N* sulphate solutions does not hold for metals in chloride solutions or in cyanide solutions. The order of any two metals may be reversed.

Polarization.—In actual practice we find that the voltages applied to a cell are in excess of those listed in Table 2; *i.e.*, it takes more than 2.55 volts to decompose a normal zinc-sulphate solution. This excess in potential is termed "polarization potential" or "polarization voltage."

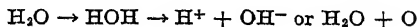
If we take a fine platinum wire as an auxiliary electrode and explore the potential drop between the cathode and anode of an electrolytic cell, we find that there is a decided drop (or rise) in potential close to either electrode surface and a relatively small drop through the solution. Ohm's law applies to the potential drop through the solution but does not apply to the over-all voltage measured between electrodes.

This excess or polarization voltage is primarily due to two causes: a change in concentration of the electrolyte next the electrodes and/or the formation of a gas film (hydrogen at the cathode, oxygen at the anode).

Concentration polarization at the cathode is due to the discharge of copper or other metal ions at a rate faster than the rate of supply of these ions to the cathode surface. Accordingly, the metal concentration of the layer of solution next the cathode decreases down to a point where other ions, notably hydrogen, are discharged in place of the metal ions and the cathode potential rises.

Conversely, at the anode the rate of solution of the anode (such as a copper anode in copper refining) proceeds faster than the copper ions formed are carried away either by ionic migration or mechanically through stirring or circulation of the solution. Consequently, the solution layer next the anode surface becomes more than saturated with copper and the electrical resistance through this layer increases and the anode potential rises.

Gas polarization is due to a film of hydrogen on the cathode or a film of oxygen on the anode. The electrical resistance of these films is higher than the electrical resistance of the original solution, consequently the electrode potentials rise. The hydrogen film as well as the oxygen film is due to the decomposition of water



In some electrolytic processes these gas films are absolutely essential. For example, chromium plating as carried out today is dependent primarily on the right thickness of an atomic hydrogen film on the cathode as elucidated by the author.¹ Films of oxygen on the anode are essential for the efficient oxidation of a large array of organic compounds.

Systems of Refining.—All refining operations may be carried on either by "multiple" or "series" arrangements. In the first, all anodes (and all cathodes) in any individual tank are in parallel, *i.e.*, they all have a direct connection to the bus bars. In the series system only the two end plates in each tank are connected to the source of current supply, one being the cathode, the other the anode. Between them are arranged as many parallel plates as is desired (up to the point where the resistance of a single tank becomes too high). The side of each plate toward the tank or end anode then becomes a cathode and receives a deposit, while the side of each plate toward the tank cathode becomes an anode and is eaten away, each plate (theoretically) moving toward the anode end of the tank by its own thickness as the refining progresses. This is treated at length under Copper Refining (page 258, of the volume entitled "Recovery of the Metals").

Tank Arrangement.—In metal refining (copper, lead, etc.) it is now very common to build tanks in long rows with a common side to each two tanks and on this to

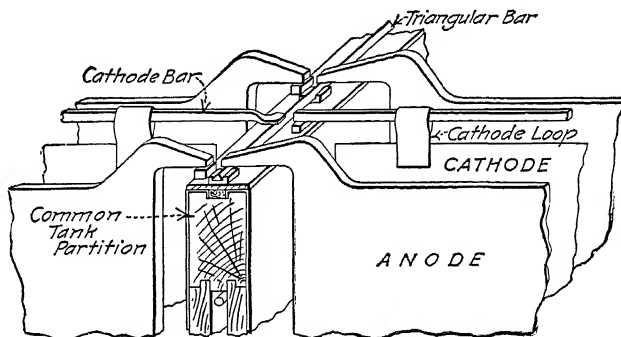


FIG. 1.—Tank and electrode arrangement.

place a triangular copper conductor. All the anode lugs of one tank rest on it and all the cathodes of the next tank. Each small section of the bar (except for the end anode or cathode) will then carry half the current of one electrode. It will be objected to at first sight that the triangular rod will not give a good contact with the electrode, but, as a matter of fact, it does, due largely to the weight and pressure of the electrodes on the bars. Furthermore, to maintain a bright metal contact the rods should be kept somewhat oily, as most tank-room atmospheres are corrosive. The thin skin of oil will not affect the contact appreciably. Figure 1 shows the arrangement of tanks and electrodes referred to.

Under Zinc Refining (page 411, of the volume entitled "Recovery of the Metals") is shown another tank arrangement that brings both the common conductor bars on one side of the tank.

As said above, the tank arrangement in an installation will be figured so as to give an economic ratio of tank to circuit resistance. The amperes to be used is a matter of dividing the ampere work to be done by the number of tanks. (One ampere will

¹ U.S. patent 1581188 (1926).

deposit 0.0011182 g. of silver per second in one cell. If put through 20 cells in series, it deposits twenty times as much.) The outside circuit will be made as short as possible, to cut down line resistance, and will be calculated to that size where the increased interest on the cost of the extra copper becomes less than the cost of the power saved.¹

Electrodes.—Both anodes and cathodes for metal refining are shown in Figs. 1 and 2. Anodes are usually supported by lugs cast on the anode (not, however, in series processes). Cathodes are supported by loops or hooks. The cathode in the multiple system is originally a starting sheet, which may be of the metal to be deposited, in which case the starting sheet is usually melted with the deposit, or it may be of other metal, and the deposited cathode metal is later stripped off.

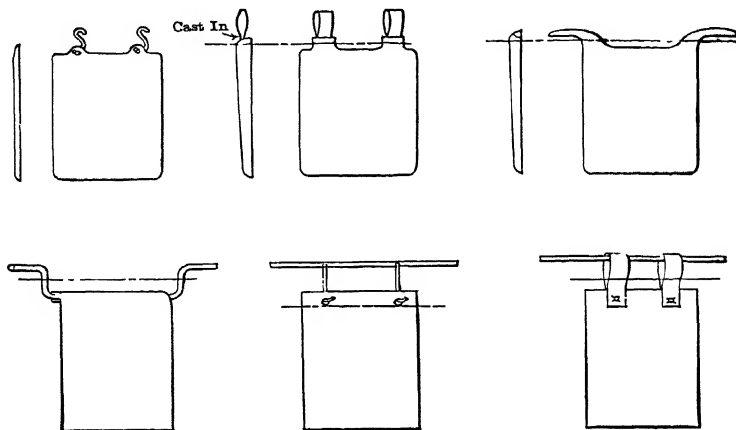
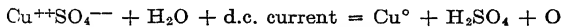


FIG. 2.—Electrode shapes.

Insoluble Anodes.—Insoluble anodes are used in all electrowinning cells, *i.e.*, in cells that are fed with leach liquors. The purpose of these cells is to extract the pure metal, such as copper or zinc, and at the same time increase the free acid content of the electrolyte by anodic reaction so that the “spent electrolyte” may be used over and over again for leaching more ore (see Chaps. XII and XIV of the volume entitled “Recovery of the Metals”).



the oxygen being formed at and evolved at the insoluble anode.

Another important application of insoluble anodes is in a certain small percentage of electrolytic cells used in the electrorefining plants where the metal content of the electrolyte of the regular cells has built up in copper (or lead or tin) beyond the specified limit for most economical cell performance. These insoluble anode cells

¹ The Institution of Electrical Engineers (British) uses a formula for calculating size of conductors which is independent of the price of copper. It is $C = 2.6A^{0.82}$ or

$$\log C = 0.82 \log A + 0.415$$

where C = current in amperes; A = area in thousandths of a square inch. While undoubtedly sizes given by this formula are safe, when copper is cheap it pays far to exceed mere safety, and it is believed that the price of copper must enter into the calculation where large currents are being carried.

are frequently called "liberator" or "equalizer" cells. These same cells function also as "complete depositing out" cells for a given fraction of the copper electrolyte which has become "foul" due to accumulation of impurities, notably nickel (or cobalt) and arsenic. This fraction of the electrolyte is electrolyzed to deposit out and recover as much good copper as possible; next, to deposit copper, with small percentages of impurities, which is sent back to the anode furnaces; and the final copper is recovered as copper sulphate by evaporation of the electrolyte. Then follows another evaporation step and nickel (or cobalt) sulfate is crystallized out.

In Table 3 are listed a number of the principal insoluble anodes in commercial use today.

TABLE 3.—GROUP A: FOR ACID SULPHATE ELECTROLYTES	
Composition	Remarks

Magnetite ^a	Limited use in Europe. Very brittle
Lead + 4 to 12% Sb ^b ...	Universally used in most electrolytic copper refineries
Duriron (13% Si).....	Limited use in copper refineries. Brittle
Copper silicide ^c	Limited use by Chile Copper Co. for making high-purity copper. Brittle
Lead + Ag ^d	Used in all electrolytic zinc plants
GROUP B: FOR MISCELLANEOUS ELECTROLYTES	
Platinum.....	For electrolytic chlorate production
Lead and lead-tin ^e	For chromium plating
Iron, nickel-plated ^f	For electrolytic H ₂ + O ₂ production
Graphitized carbon.....	In over 90% of the alkali-chlorine cells
Lead-silver.....	In a few alkali-chlorine plants in Europe
Lead-tin-Sb-Co ^g	In the Knoxville plant for electrolytic manganese

* The magnetite anodes as used in Germany and Italy are hollow and closed at the lower end. A heavy copper plate on the inside wall counteracts the high electrical resistance of magnetite. Aside from being decidedly brittle, these anodes dissolve slowly, introducing objectionable iron into the electrolyte.

^b The antimony was originally added to the lead to counteract sagging or "creep." But it also adds to its insolubility. Antimonial lead cannot be used in the electrolytic zinc cells; for a trace of antimony getting into the electrolyte seriously interferes with the electrodeposition of zinc. In these cells, lead-silver anodes are used instead (see below).

* The copper silicide anode has been in use at Chuquicamata for 20 years. Fink, U.S. patents 1441567, 1441568 (1923). It is now being largely replaced by a lead-antimony-silver anode.

^d See footnote b; also TAINTON, *Trans. A.I.M.E.*, 1929, p. 192; FINK and PAN, *Trans. Electrochem. Soc.* Vol. 46, p. 349, 1924; *ibid.*, Vol. 49, p. 85, 1926. See also C. G. Fink and R. W. Low, U.S. patent 1740291 (1929).

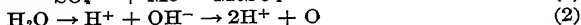
* C. H. Eldridge, U.S. patent 1975227 (1934).

^f SUTHERLAND, *Trans. Electrochem. Soc.*, Vol. 85, p. 231, 1944.

* C. G. Fink and M. Kolodney, U.S. patent 2320773 (1943); C. L. Mantell, U.S. patent 2340400 (1944). With ordinary lead anodes or lead-antimony anodes, about half the manganese is deposited on the anode (as MnO_2) and half (as metal) on the cathode; with the above anode, practically all the manganese is deposited on the cathode.

The theoretical basis of the insolubility of an anode was laid down by the writer some twenty years ago and has been found to hold good in the case of every new insoluble anode developed.

At the anode surface we are dealing primarily with two fundamental reactions: (1) The attack on the metal(s) of the anode by the discharged anion, such as SO_4^{--} , causing the metal(s) to go into solution; and (2) the discharge of OH^- ions (supplied by the water) to form oxygen.



It will be appreciated that the more readily reaction (2) takes place as against reaction (1), the more insoluble will be the anode. Now, it so happens that as a rule the evolution of oxygen at the anode is retarded because of the stubborn adhesion of an oxygen film to the surface of the anode. The degree of adhesion varies with different metals and with different physical states of the metal (oxygen is more easily evolved from a platinum-black anode than from a bright, polished platinum anode). Anything we can do to weaken or destroy this adhesion of the gas film will favor the evolution of oxygen and simultaneously lower the voltage at which oxygen will be evolved at the anode. In other words, the lower the oxygen over-voltage, the more easily and the more rapidly will oxygen be evolved under a given set of conditions.

Among the many anode surfaces investigated by the writer, the following are a few of those that show exceptionally low oxygen film adhesion, or low oxygen over-voltage: Ni_2O_3 , Co_2O_3 , MnO_2 , PbO_2 , Ag_2O , SnO_2 , and, this is important, combinations of these oxides. These oxides might be termed "catalysts."

A pure copper plate can be rendered insoluble anodically in an acid copper-sulphate solution by applying a tight-fitting film of a combination of the above oxides to the anode surface.

These oxides have thus made it possible so to lower the anode potential below the minimum voltage at which anode metal will go into solution that the current efficiency for oxygen production is very nearly 100 per cent. Frequently, before putting newly cast anodes into use, it is necessary to "form" them in a special cell. In other words, a preliminary electrolyzing is necessary at times to form a properly functioning catalytic surface on the anode.

Cathode Materials.—Ordinarily, in metal refining, the cathode will be the same material as the metal deposited. This is not true in zinc refining, simply because there is a decided tendency to corrosion of the cathode at the solution line (where the oxygen of the air helps the electrolyte in dissolving zinc), and if zinc cathodes were used they would cut through and drop down into the tank, causing short circuits.

Nor are silver cathodes used in the Thum-Balbach silver-refining process, as it would tie up too much silver and the deposit would be less readily removed than from the graphite plate or stainless-steel cathode actually used. In zinc electrolysis it would be more convenient to have an all-zinc cathode which could be melted entirely, rather than have to have to strip the zinc from the aluminum cathodes. But the labor cost of stripping is less of a disadvantage than those evils ensuing from "cut" cathodes. It is again a balancing of economic factors, the ability to do which, after all, is what differentiates the chemical engineer from the chemist.

In multiple copper refining, the metal is deposited on "starting sheets," which are thin copper sheets that have been deposited on greased copper or hard lead plates and then stripped off. Starting-sheet making is given more fully on page 258. Vol. II.

What is said of copper starting sheets can be considered typical. Lead and tin sheets are made by casting them on an inclined iron table. In nickel refining, iron-plate cathodes are often employed and the nickel deposit mechanically removed.

Commercial Electrolytes (Refining and Winning).—The cheapest acid equipment used in metallurgical and chemical engineering is made of lead: lead-lined tanks, pumps, launders, etc. Accordingly, when presented with the problem of carrying out a new electrolytic metal process on a commercial scale, the engineer first carefully considers the possibility and feasibility of using an acid sulphate electrolyte together with lead equipment. Most commercial electrolytes are sulphate solutions: copper, zinc, cadmium, nickel, tin, cobalt, manganese.

Copper.—Both the electrowinning and refining baths are the acid sulphate: about 3 to 3¾ per cent copper and 15 to 17 per cent free sulphuric acid.

Zinc.—The zinc-winning electrolyte carries 120 g. per l. of zinc entering the electrolytic cells and 50 g. per l. of zinc leaving the cells.

Cadmium.—Cadmium is a by-product of electrolytic zinc. The cadmium cells are fed with solution running 100 to 200 g. per l. of cadmium. The discharge of the cells carries 56 to 140 g. per l. of free H_2SO_4 .

Nickel.—The nickel-refining solution contains 40 g. per l. of nickel as sulphate. Besides sulphuric acid the solution contains 20 g. per l. of boric acid.

Tin.—The tin refining bath contains 100 g. per l. of tin and 40 g. per l. of free sulphuric acid. Cresol-phenol sulphonic acid is used as addition agent.

Cobalt.—The cell feed of the cobalt-winning electrolyte contains about 20 g. per l. of cobalt as cobalt sulfate, 50 g. per l. of boric acid and 5 g. per l. of sodium fluoride.¹

Manganese.—In the electrolytic winning of manganese from low-grade ores the electrolytic cell feed contains 25 g. per l. of manganese as sulphate. The pH of the solution is about 7.4.

Lead.—Electrolytic refining of lead is carried out in order to recover the bismuth present as an impurity. The refining solution contains 75 g. per l. of lead as lead fluosilicate, PbSiF_6 , and 145 g. per l. of free fluosilicic acid.

Bismuth.—There are two competing methods for the recovery of bismuth from impure lead: the Betts and the Betterton. In the Betts process (see above), the insoluble slimes contain the bismuth. These are treated and finally cast into anodes. Pure bismuth metal is produced electrolytically from these anodes by electrolysis. The solution contains 100 g. per l. of hydrochloric acid and 4 g. per l. of bismuth. [In the Betterton process, calcium (or magnesium) metal is added to the impure molten lead and a calcium-bismuth dross is formed.]

Silver.—The solution usually employed in the electrolytic parting of silver from gold contains 50 to 60 g. per l. of silver, as silver nitrate, and 40 g. per l. of copper, as nitrate. Free acid is to be avoided lest the silver deposit be contaminated with palladium. The presence of the copper is "essential" since this prevents the formation of long needlelike silver crystals which give rise to short circuits through the canvas diaphragm. The cathode is either graphite (Thum cell) or stainless steel (Moebius cell).

Gold.—The insoluble black slimes from the silver parting cell (see above) are carefully washed and then melted and cast into anodes. Their composition varies from plant to plant. There are present, besides gold, a little silver, platinum, and palladium. The gold-refining electrolyte contains 90 g. per l. of gold as chloride and 110 g. per l. of free hydrochloric acid. Any silver in the anode is precipitated as silver chloride, collecting in the bottom of the cell. *Platinum and palladium* accumulate in the electrolyte and are chemically removed by adding ammonium chloride to a portion of the electrolyte. $(\text{NH}_4)_2\text{PtCl}_6$ and $(\text{NH}_4)_2\text{PdCl}_6$ are collected on a filter. The two metals are chemically separated (palladous chloride is not precipitated by NH_4Cl).

Tellurium is a by-product of copper refining. The impure metal recovered from the Doré slags and from the Cottrell precipitator is refined electrolytically from an alkaline sodium tellurate bath.

Electroplating.—During the last twenty-five years revolutionary, far-reaching changes have occurred in the electroplating field. Electroplating has been promoted from a trade to a profession. This happened when control of the plating industry passed into the hands of the automotive industry. Where 1-gal. plating cells had been used in the shop, 1000-gal. tanks were used in the automobile factories. Another revolutionary change is the preplating of steel and other metal strip before fabricating, whereas for hundreds of years the plating step was the last, all fabricating being com-

¹ CHURCHWARD, SHELTON, KNICKERBOCKER, *Trans. Electrochem. Soc.*, Vol. 85, p. 245, 1944.

pleted before plating. Plating rapidly moving strip metal such as strip steel involves cathode current densities of 2000 or 3000 amp. per sq. ft. as against 10 or 15 amp. per sq. ft. when plating articles in the old plating shop. Control of the plating industry is now passing into the hands of the steel companies.

To assure perfect adhesion, the metal surface to be plated must be free from grease and other impurities. Nowadays electrolytic cleaning in an alkali bath is usually resorted to, the article to be plated being made cathode. Heavy furnace scale on steel is likewise blasted off electrolytically, a little lead or tin salt being added to an acid sulphate bath (Bullard-Dunn process). The lead or tin eliminates pitting during electrolytic cleaning.

In the production of the familiar phonograph disk records, five separate electroplating steps are involved. The original recording is on a soft wax plate. The wax surface is made electrically conductive and an electroplate of copper applied. Then follow four more electroplating steps. There is no other product in the entire electrodeposition field that demonstrates so forcibly the remarkable exactness with which an electrodeposit reproduces the minutest detail of the base to which the deposit is applied. Any imperfection in a record, no matter how slight, would be detected by a normally sensitive ear.

Lately a radical innovation has been introduced in the mode of treating a metal surface preparatory to electroplating it. It has been found in the case of aluminum that a carefully oxidized aluminum surface (by anodizing) gives better adhesion than an aluminum surface treated with acids and alkalis to remove the oxide film. This was a surprise to the electrolytic-zinc engineer as he depends on poor adhesion of zinc to the aluminum cathode for ease of stripping.

Similarly, better adhesion of silver to steel is obtained after phosphating (Parkerizing) the steel surface than by carefully preparing a virgin steel surface.

The Anaconda Copper Co. has developed a process of making endless paper-thin copper strip by plating copper on a revolving greased lead drum (5 ft. in diameter).

The most important metals electroplated today¹ are listed in Table 4.¹

TABLE 4.—METALS COMMERCIALY ELECTROPLATED

Metal	Baths used	Remarks
Cadmium.....	Cyanide and sulphate	Fair protection for steel
Chromium.....	Chromic acid and catalyst	Excellent wear resistance. Decorative
Copper.....	Acid sulphate Cyanide	Electroforming; Copper plating steel
Gold.....	Cyanide	Jewelry
Iron.....	Chloride and sulphate	Electroforming; electro-winning
Nickel + X% Co..	Acid sulphate	Undercoat for chromium on steel, brass, and die castings.
Palladium.....	Cyanide base	Jewelry
Rhodium.....	Acid sulphate	Searchlight reflectors; jewelry
Silver.....	Cyanide	Tableware
Tin.....	Acid sulphate Alkali stannate	On steel, copper, etc.
Zinc.....	Acid sulphate and cyanide	On steel

¹ For details see "Modern Electroplating," The Electrochemical Society, Inc., 1942.

Deposition of Alloys.—Aside from plating the metals listed in Table 4, alloys of these and other metals may be deposited on a commercial scale. For example, brass plating steel has been in commercial production for 25 years. Many hardware articles that look like solid brass or bronze are plated steel articles. The decided success in brass plating from a bath containing the noble metal, copper, and the relatively base metal, zinc, gave investigators much encouragement to try depositing other alloys.

Today a number of alloys are being electrodeposited commercially: nickel and cobalt, nickel and copper, nickel and iron, lead and tin,¹ and others.² Variables such as metal A vs. metal B concentration, bath temperature, complex salt formation, degree of alkalinity or acidity, voltage and current density, and degree of agitation are so altered and regulated that the deposition potential of both metals is identical. Thus, for example, in the case of the brass bath the higher the current density, the lower the copper content of the alloy deposit; and the higher the bath temperature, the higher the copper content. The changes in ratio of copper to zinc in the cathode deposit are readily detected by sight. The deposit can be made to pass from a red copper color through a golden brass color to an almost pure white zinc color.

Electrolysis of Fused Salts.—Although the fused electrolyte industry is not nearly so diversified as the aqueous electrolyte industry, the products are fully as important, if not more so. The chief products are sodium metal, magnesium, calcium, and aluminum. Other metal products include potassium, strontium, barium, cerium, lanthanum, tantalum, columbium, and tungsten.

Whereas the metal sulphates are the predominant salts in aqueous electrolytic processes, the metal chlorides are the preferred salts in fused electrolyte processes. The chlorides (or fluorides) are more stable than other salts, have lower melting points, and are better conductors of electricity. The conductivity of these fused salts is much higher than that of their aqueous solutions. The ionization is almost 100 per cent. Faraday's laws apply rigidly. Complexes are seldom formed. The speed of the ions is largely affected by the temperature. The decomposition voltage is usually about 2 or 3 volts. The following very simple relation holds: molar conductivity \times temperature coefficient = constant for any one particular temperature. No such simple relations hold for the concentrated aqueous solutions used in refining and winning. The melting point of a fused salt is readily reduced by the addition of a second or third salt. This is a universal practice. Similarly, the conductivity of the molten mass can be increased or decreased by the addition of a second or third salt. The cathode deposit when solid is crystalline. But usually the temperature of the fused bath is above the melting point of the metal, and consequently the cathode deposit is a liquid (Al, Mg, Na, etc.)

Current efficiencies are usually lower than with aqueous electrolytes. This is largely due to the following causes: (1) volatilization of the deposited metal, (2) chemical side reactions due to rapid diffusion of electrode products, (3) formation of metal fog within the molten salt, (4) reaction of the metal produced with the oxygen of the surrounding atmosphere. In spite of all these factors, years of development have resulted in improved fused-salt cell operation with current efficiencies frequently above 80 per cent. As to energy efficiencies, these are usually one-half of the values attainable in aqueous electrolytes. Even so, the price per cubic foot of the fused-cell metal products is decidedly less than that of any of the aqueous electrolyte metal products. On the volume basis, blast-furnace iron is the cheapest metal, next comes sodium, then magnesium, and then aluminum. The price of copper or nickel is ten times that of aluminum on a volume basis.

¹ HERMSDORF and HEBERLEIN, *Tech. Paper*, 680, *Trans. A.I.M.E.*, Vol. 121, p. 289, 1936.

² See "Modern Electroplating," The Electrochemical Society, Inc., New York, 1942.

Raw Materials for Fused Electrolytes.—In general, the raw-material supply for the chief fused-electrolyte products is practically infinite. In the case of aluminum we have the bauxites, alunites, and clays; in the case of sodium, magnesium, and calcium, the vast mineral deposits, and the salts of the oceans. This inexhaustible raw-material supply is in strong contrast with the serious early depletion of ores of copper, zinc, nickel, and other so-called "common" metals, estimated at within 40 or 50 years at the rate of consumption of 1930-1935.

Fused-electrolyte Cell Problems.—Whereas with aqueous electrolytes a large variety of cell-construction materials are in use (concrete, wood, metal, earthenware, and glass), with fused salts we are very limited in choice. Acheson graphite has been the "lifesaver" of the fused-electrolyte industry. The Hall-Héroult cell is made of graphite plates held in place by iron; the anodes are graphite which contribute very materially to the reduction of the aluminum. Graphite anodes are used in all the fused-salt processes. The body of the cell is made of iron whenever possible.

A very disturbing factor during electrolysis of fused fluorides is the well-known anode effect. In the aluminum bath, a high-resistance gas film (usually fluorine) covers the anode surface below the liquid salt line and is a direct indication that the aluminum oxide content is almost depleted. Addition of new aluminum oxide will cause the cell voltage to drop back to normal (6.0 volts). In the cell used for the production of fluorine gas from a molten HKF_2 bath, the anode effect is due to moisture in the salt and the high-resistant gas film is largely carbon monoxide. Here drying the salt and raising the temperature slightly will eliminate the high potential. CO is evolved from a graphite anode at elevated temperatures more readily than at lower temperatures. The reverse holds true for fluorine gas.

Power Requirements, Electrochemical Industries.—On page 627 and on this this will be found tables of power requirements for the production of various chemicals and refined metals by use of the electric current.

POWER REQUIREMENTS—ELECTROTHERMAL INDUSTRIES

MATERIAL	KILOWATT HOURS PER TON	MATERIAL	KILOWATT HOURS PER TON
Aluminum.....	24,000	Ferrosilicon (75 per cent).....	10,000
Alundum.....	2,000	Ferrotungsten (70 per cent)....	7,600
Barium oxide.....	1,200	Ferrouanium (40 per cent)....	8,000
Cadmium.....	2,500	Ferrovandium (35 per cent)....	6,800
Calcium carbide.....	4,000	Graphite.....	7,800
Calcium cyanamide.....	3,750	Iron (electrothermic).....	2,500
Carbon bisulphide.....	850	Iron (electrolytic).....	4,000
Carborundum.....	8,500	Lead.....	145
Caustic soda, 2,000 lb.....	3,000	Magnesium.....	27,000
Chlorine, 1,760 lb.....		Nitric acid.....	17,500
Copper (electrolytic refined)....	300	Phosphorus.....	12,000
Copper (electrolytic).....	2,600	Potassium chlorate.....	1,350
Ferrochromium (60 per cent)....	8,000	Sodium.....	20,000
Ferromanganese (76 per cent)....	5,000	Sodium chlorate.....	7,000
Ferromolybdenum (60 per cent).	8,400	Tin.....	175
Ferrosilicon (50 per cent).....	5,000	Zinc.....	4,000

From *General Electric Rev.*, June, 1921, with some corrections (1943).

APPENDIX

METRIC—ENGLISH EQUIVALENTS

Length		Volume	
1 ft.	= 0.30480060096 m.	1 cu. yd.	= 0.764559445 cu. m.
1 in.	= 2.540005 cm.	1 cu. ft.	= 0.028317 cu. m.
1 m.	= 3.28083 ft. = 39.370000 in.	1 cu. in.	= 16.3872 cc.
		1 cu. ft.	= 28.316 l.
		1 cu. in.	= 16.3867 ml.
		1 cu. m.	= 1.3079 cu. yd.
		1 cc.	= 0.06102 cu. in.
		1 l.	= 61.025 cu. in. = 0.035315 cu. ft.
Area			
1 sq. in.	= 6.452 sq. cm.		
1 sq. ft.	= 0.09290 sq. m.		
1 sq. yd.	= 0.83613 sq. m.		
1 sq. m.	= 1.1960 sq. yd. = 10.764 sq. ft.		
1 sq. cm.	= 0.15500 sq. in.		
Mass			
1 lb. (avoirdupois)	= 0.4535924217 kg.		
1 oz. (avoirdupois)	= 28.3495 g.		
1 oz. (troy)	= 31.10348 g.		
1 dram (℥ = 3 ℥, apoth.)	= 3.887935 g.		
1 kg.	= 2.20462234 lb. (avoirdupois)	= 2.67923 lb. (Troy)	
1 g.	= 0.035274 oz. (avoirdupois)	= 0.032151 oz. (Troy)	= 15.4324 grains
Capacity			
1 qt. (liquid)	= 0.94633307 l.		
1 qt. (dry)	= 1.1012 l.		
1 l.	= 1.05671 qt. (liquid)	= 0.9081 qt. (dry)	= 33.8147 fl. oz.
1 fl. oz.	= 0.0295729 l.	= 1.80469 cu. in.	
NOTE.—1 gal. (liquid)	= 231.0 cu. in.		
1 bu. (dry)	= 2,150.42 cu. in.		
1 l.	= 1,000.027 cc.		
Energy			
1 hp.	= 0.746 kw. = 33,000 ft.-lb. per min.		
1 kw.	= 1.341 hp. = 1,000 joules per sec.		
1 ft.-lb.	= 1.383×10^7 ergs = 1.383 joules = 0.1383 kg.-m.		
1 poundal	= 13,825 dynes.		
1 gram's weight	= 980 dynes.		
1 pound's weight	= 444,518 dynes.		
1 hp.-year	= 6,535 kw.-hr.		
1 kw.-yr.	= 11,747 hp.-hr.		

INTERNATIONAL ATOMIC WEIGHTS, 1943

Element	Symbol and atomic number	Weight	Valence ¹	Electro-chem. equivalents, g. per amp.-hr.	Melting points, deg. C.	Boiling points deg. C., visible ebullition
Aluminum.....	Al(13)	26.97	3	0.3355	658.7	1800.0
Antimony.....	Sb(51)	121.76	3	1.5145	630.0	1460.0
Argon.....	A(18)	39.9	0	-189.3	-186.0
Arsenic.....	As(33)	74.91	3	0.9318	850.0	450.0 ²
Barium.....	Ba(56)	137.36	2	2.5619	850.0	
Beryllium.....	Be(4)	9.02	2	0.1683	1285.0	
Bismuth.....	Bi(83)	209.0	3	2.5992	271.0	1440.0
Boron.....	B(5)	10.82	3	2350.0	3500 ²
Bromine.....	Br(35)	79.92	1	2.9814	-7.3	58.75
Cadmium.....	Cd(48)	112.41	2	2.0974	320.9	778.0
Caesium.....	Cs(55)	132.91	1	26.0	
Calcium.....	Ca(20)	40.08	2	0.7478	810.0	
Carbon.....	C(6)	12.01	4	0.1118	>3800.0	3700.0
Cerium.....	Ce(58)	140.13	3	1.7426	623.0	
Chlorine.....	Cl(17)	35.46	1	1.3230	-101.5	-37.6
Chromium.....	Cr(24)	52.01	3	0.6470	1520 to >Fe	2200.0
Cobalt.....	Co(27)	58.94	3	1.1000	1610 ²	
Columbium.....	Cb(41)	92.91	5	1950-2200	
Copper.....	Cu(29)	63.57	2	1.1858	1083.0	2100.0
Dysprosium.....	Dy(66)	162.46	3			
Erbium.....	Er(68)	167.2	3			
Europium.....	Eu(63)	152.0	3			
Fluorine.....	F(9)	19.0	1	0.7085	-223.0	-187.0
Gadolinium.....	Gd(54)	156.9	3			
Gallium.....	Ga(31)	69.72	3	30.1	
Germanium.....	Ge(32)	72.6	4	958.0	
Gold.....	Au(79)	197.2	3	2.4522	1063.0	2100.0
Hafnium.....	Hf(72)	178.6				
Helium.....	He(2)	4.003	0	-271.9	-268.8
Helvetium.....	Hv(85)					
Holmium.....	Ho(67)	164.94				
Hydrogen.....	H(1)	1.008	1	0.03761	-259.0	-252.8
Illinium.....	Il(61)					
Indium.....	In(49)	114.76	3	1.4271	154.5	1000.0
Iodine.....	I(53)	126.92	1	4.7353	114.0	184.35
Iridium.....	Ir(77)	193.1	4	2300.0	2850.0
Iron.....	Fe(26)	55.85	2	1.0416	1530 ± 5	2450.0
Krypton.....	Kr(36)	83.70	0	-169.0	-151.7
Lanthanum.....	La(57)	138.92	3	810.0	
Lead.....	Pb(82)	207.21	2	3.8651	327.4	1525.0
Lithium.....	Li(3)	6.94	1	0.2589	186.0	500.0
Lutecium.....	Lu(71)	174.99	3			
Magnesium.....	Mg(12)	24.32	2	0.4536	651.0	1120.0
Manganese.....	Mn(25)	54.93	2	1.0255	1260 ± 20	1900.0
Masurium.....	Ma(43)					
Mercury.....	Hg(80)	200.61	2	7.4840	-38.87	357.0
Molybdenum.....	Mo(42)	95.95	2	1.7903	2550.0	3350.0
Neodymium.....	Nd(60)	144.27	3	840.0	
Neon.....	Ne(10)	20.18	0	-253.0	-245.9

¹ In those cases in which a metal has two valences, the valence given corresponds to the electro-chemical equivalent, and may not necessarily be the commoner one.

² Sublimes. ³ Commercial metal, about 1480°C.

INTERNATIONAL ATOMIC WEIGHTS, 1943.—(Continued)

Element	Symbol and atomic member	Weight	Valence ¹	Electro-chem. equivalents, g. per amp.-hr.	Melting points, deg. C.	Boiling points deg. C., visible ebullition
Nickel.....	Ni(28)	58.69	2	1.0951	1452 ± 3	2450.0
Nitrogen.....	N(7)	14.008	3	0.1742	-210.5	-195.7
Osmium.....	Os(76)	190.2	4	2700.0	2950.0
Oxygen.....	O(8)	16.00	2	0.2985	-218.0	-183.0
Palladium.....	Pd(46)	106.7	2	1.9903	1550.0	2540.0
Phosphorus.....	P(15)	30.98	44.1	287.0
Platinum.....	Pt(78)	195.23	4	1.8206	1755.0	2650.0
Polonium.....	Po(84)	214.2	4	1.9586
Potassium.....	K(19)	39.096	1	1.4590	62.3	667.0
Praseodymium.....	Pr(59)	140.92	3	940.0
Protactinium.....	Pa(91)	231	5
Radium.....	Ra(88)	226.05	2	900.0
Radon.....	Rd(86)	222.4	0	-71.0	-62.0
Rhenium.....	Re(75)	186.31	4
Rhodium.....	Rh(45)	102.91	3	1.2797	1940.0	2750.0
Rubidium.....	Rb(37)	85.48	1	38.0	696.0
Ruthenium.....	Ru(44)	101.7	4	>1950.0	2780.0
Samarium.....	Sa(62)	150.43	3	1350.0
Scandium.....	Sc(21)	45.10	3	1200.0(?)
Selenium.....	Se(34)	78.96	2	1.4733	218.5	690.0
Silicon.....	Si(14)	28.06	4	0.2618	1420.0	3800.0
Silver.....	Ag(47)	107.88	1	4.0245	961.0	1955.0
Sodium.....	Na(11)	22.997	1	0.8579	97.5	742.0
Strontium.....	Sr(38)	87.63	2	1.6333	>Ca <Ba
Sulphur.....	S(16)	32.06	2	0.5980	122.8-119.2	444.5
Tantalum.....	Ta(73)	180.88	5	2850.0
Tellurium.....	Te(52)	127.01	2	2.3803	451.0	1390.0
Terbium.....	Tb(65)	159.2	3
Thallium.....	Tl(81)	204.39	1	7.6249	302.0	1700.0 ²
Thorium.....	Th(90)	232.12	4	2.1649	>1700.0 <Pt
Thulium.....	Tm(69)	169.4	3
Tin.....	Sn(50)	118.7	2	2.2141	231.9	2270.0
Titanium.....	Ti(22)	47.9	4	0.4490	1795.0 ± 150.	2700.0
Tungsten.....	W(74)	183.92	6	1.1493	3267.0	3700.0
Uranium.....	U(92)	238.07	4	Near Mo.	3100.0
Vanadium.....	V(23)	50.95	5	1720.0 ± 20.0
Xenon.....	Xe(54)	131.3	0	-140.0	-109.1
Ytterbium.....	Yb(70)	173.04	3	1800.0(?)
Yttrium.....	Yt(39)	88.92	3	1490.0
Zinc.....	Zn(30)	65.30	2	1.2199	419.4	918.0
Zirconium.....	Zr(40)	91.22	4	1700.0(?)

NOTE.—The hypothetical coronium supposed to exist in the solar corona has lately been shown to be highly ionized nickel, calcium and iron.

¹ In those cases in which a metal has two valences, the valence given corresponds to the electrochemical equivalent, and may not necessarily be the commoner one.

² Also given as 1280°C.

THE ATOMIC NUMBERS

(The arrangement is like that of a logarithmic table. The first figure of the number is given by column 1, the second figure by the numbers at the tops of the columns.)

	0	1	2	3	4	5	6	7	8	9
0	..	H	He	Li	Be	B	C	N	O	F
1	Ne	Na	Mg	Al	Si	P	S	Cl	A	K
2	Ca	Sc	Ti	V	Cr	Mn	Fe	Co	Ni	Cu
3	Zn	Ga	Ge	As	Se	Br	Kr	Rb	Sr	Y
4	Zr	Cb	Mo	Ma	Ru	Rh	Pd	Ag	Cd	In
5	Sn	Sb	Te	I	Xe	Cs	Ba	La	Ce	Pr
6	Nd ¹	¹	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm
7	Yb	Lu	Hf	Ta	W	Re	Os	Ir	Pt	Au
8	Hg	Tl	Pb	Bi	Po(?)	²	Rd	³	Ra	Ac(?)
9	Th	Pa	U							

¹ Named illinium by the discoverers, B. S. Hopkins, J. A. Harris, and L. F. Yntema. Probably Rolla's florentium.

² Possibly Helvetium—Dr. Walther Minder.

³ Possibly Virginium—Dr. F. Allison.

EQUILIBRIUM CONDITIONS FOR SULPHATES

$\text{Fe}_2(\text{SO}_4)_3 \rightleftharpoons \text{Fe}_2\text{O}_3 + 3\text{SO}_3$		$\text{Al}_2(\text{SO}_4)_3 \rightleftharpoons \text{Al}_2\text{O}_3 + 3\text{SO}_3$		$2\text{CuSO}_4 \rightleftharpoons 2\text{CuO} \cdot \text{SO}_3 + \text{SO}_3$		$2\text{CuO} \cdot \text{SO}_3 \rightleftharpoons 2\text{CuO} + \text{SO}_3$		$\text{ZnSO}_4 \rightleftharpoons \text{ZnO} + \text{SO}_3$	
Temp- erature	Milli- meters	Temp- erature	Milli- meters	Temp- erature	Milli- meters	Temp- erature	Milli- meters	Temp- erature	Milli- meters
553°	23	572°	28	546°	43	600°	62	675°	5
570°	33	621°	51	588°	55	653°	98	690°	6
592°	45	681°	120	615°	70	686°	123	720°	24
614°	70	702°	180	642°	98	705°	139	750°	61
634°	113	720°	261	665°	130	728°	173	775°	112
650°	149	731°	356	700°	233	745°	209	800°	189
660°	182	742°	480	714°	324	775°	298		
680°	286	748°	692	725°	460	805°	542		
690°	401	731°	647				
699°	560								
707°	715								

Temperature in degrees centigrade.

INDEX

A

Acheson graphite electrodes, 608, 628
 Acid sulphate electrolytes, 637
 Activating reagents, 156
 Addition agents (flotation), 156
 Admiralty metal, 503
 Admic ("admiralty" nickel), 514, 564
 Adobe roasting furnace, 288
 Agglomerating (*See* Briquetting; Nodulizing; Sintering)
 Air cleaning, 395
 Air lifts, 434
 Akins classifier, 116
 Allen cone classifier, 110
 Alloy steels, 482, 489, 547, 550
 Alloys, deposition of, 641
 Alternating-current electric furnaces, 587
 Altitude, effect on compressors, 448
 Alumina brick, 369
 refractories, 362
 Alumina-silica equilibrium diagram, 352
 Aluminum alloys, 517, 568
 Aluminum bronzes, 506
 Aluminum chips, briquetting, 414
 Aluminum, in construction, 515
 properties of, 517, 568
 Ambrac, 513, 516
 American filter, 204
 Amines in flotation, 155
 Amorphous-carbon resistors, 608, 627
 Ampere equivalents, 631
 Angle of nip, 21
 Anodes, insoluble, 636, 637
 shapes, 636
 Antimonial lead, 562
 Applied electrochemistry, 629
 Arc furnaces, 594
 Arsenides and antimonides, roasting, 283
 Ascology, 554, 555
 Aterite alloy, 564
 Atomic numbers, 646
 Atomic weights, 644, 645
 Automatic temperature control, 271

B

Baffle separators, 399
 Bag filter, automatic, 400
 Ball mills, 43, 48
 Bartlett-Hayward Feld-type scrubber, 402
 Basic refractories, 370
 burned magnesite brick, 371
 chrome, 372
 dolomite, 372, 374
 magnesite, 370
 metalkase, 372
 Bassanese furnace, 612
 Batch drier, 12
 Beams and girders, 456, 492, 532
 Bearing bronzes, 510
 metals, 509
 Bending, beams, 457
 Bending, columns, 459
 Benedict's alloy, properties, 513
 Binding materials, for briquetting, 424-427
 Bins, construction of, 468
 Bird centrifugal separator, 213
 Bismuth electrolyte, 638
 Black-body and nonblack-body conditions, 258, 266
 Blast roasting, applications of, 322
 Carmichael-Bradford process, 304, 306
 chemistry of, 305
 definition of, 301
 Huntington-Heberlein process, 303, 306
 Savelsbury process, 304, 306
 Blast apparatus for chimneys, 441
 Blende roasting, 283
 Blowers, 452, 454
 Blowing, separation by, 146
 Brass pipe, in construction, 499, 502
 Brass rod in metallurgical construction, 501, 502
 Brasses and bronzes, 500, 557

- Brassert and Steinbart tower washers, 401
 - Brick in metallurgical construction, 534
 - specifications, 534, 535, 536
 - Brickwork, estimating, 385
 - Briquette plant design, 415
 - Briquetting, aluminum chips, 414
 - binders in, 424
 - cast-iron borings, 413
 - copper alloys, 413
 - definitions, 409
 - exothermic briquettes, 428
 - metal scrap, 409
 - securing bond, 414
 - Brittleness in metals, 2
 - Bronzes in construction, 557, 560
 - Brown straight-line roasting furnace, 290
 - Brückner and Howell-White revolving-cylinder roasters, 299
 - Brunton sampler, 64
 - Buhrstone mills, 37
 - Buildings, Ketchum's specifications for, 467
 - Bundling or cabbaging, 414
 - Burt filter, 197
- C
- Cable connections to furnaces, 605
 - Cadmium alloys in construction, 520
 - Cadmium electrolyte, 639
 - Calibrating thermometers and pyrometers (table), 232
 - Calite as construction material, 551
 - Caldecott cone classifier, 107
 - Callow cone classifier, 107
 - Capacitors, 597
 - Carbide and silicide furnaces, 619
 - Carbon nipples, physical properties, 626
 - standard sizes, 628
 - Carbon electrodes, 606
 - apparent density, 607
 - chemical composition, 607
 - resistivity, 607
 - resistor carbon and graphite, 607
 - trueness to form, 607
 - Carbon steels, properties, 479-481
 - Carborundum, furnace for producing, 620
 - Carmichael-Bradford process, 304, 306
 - Carpenter centrifugal separator, 213
 - Cascade (Hardinge) mill, 51
 - Casting brass in construction, 500
 - Cast iron, in construction, 474, 545, 546
 - Cast steels, properties, 548
 - Cathode materials, 638
 - shapes of, 636
 - Cazo process, 279
 - Cell voltage, 633
 - Cements, blended, 525
 - natural, 525
 - portland, 525
 - puzzolan, 525
 - Centrifugal fans, 442
 - Centrifugal pumps, 438
 - Centrifugal separation, 213
 - Changing filter cloths, 202
 - Characteristic curves, 450
 - Charcoal, 219
 - Chemical stoneware in metallurgical construction, 570
 - Chilean mills, 36
 - Chimneys, 438, 441
 - horsepower of, 441
 - Chloridation roasting, 279
 - Chloridizing roasting, 279
 - Chrome-nickel alloys in construction, 512, 567
 - Chrome-nickel steels, 551
 - Chrome-silicon steels, 553
 - Chrome steels in construction, 485, 549, 551
 - Chromel, properties, 567
 - Circular furnaces, for roasting, 292
 - Classification, definition, 101
 - history, 101
 - settling ratios, 105
 - Classifiers, drag belt, 120
 - free settling, 120
 - hindered settling, 122
 - mechanical, 111
 - nonhydraulic, 107
 - spigot, 107
 - surface-current, 107
 - types, of, 102
 - Clays, firing behavior of, 353
 - Coal, analyses, 215
 - classes, 214
 - sampling, 66
 - storage, 228
 - Cobalt electrolyte, 639
 - Coke, 215
 - low temperature distillation, 218
 - shatter test, 217
 - Collectors (flotation), 153
 - Colloidal fuel, 228

The subject matter of the other volume is not indexed here

- Columns, in construction, 494
- Combined stress in construction, 462
- Combustion, 220
 - data on, 222
 - heats of, 223
 - losses in, 224
 - maximum temperatures, 223, 224
- Compressed-air pumping, 433
- Compressors, 449-454
 - centrifugal, 449, 451
 - cooling, 451
 - effect of altitude, 448
 - efficiency, 449
 - hydraulic, 446
 - leakage, 451
 - power required, 451
- Concentration, by blowing, 146
 - electromagnetic, 144
 - electrostatic, 144
 - fine sand, 137
 - free-settling velocities, 128
 - hand sorting, 127
 - heavy medium, 127
 - jigging, 129
 - limits of, 127
 - magnetic separation, 143
 - sand tables, 138
 - slime treatment, 142
 - testing, 126
 - vanners, 142
 - washers, 127
- Concrete, in metallurgical construction, 520, 525
 - design of, 527, 530
 - proportioning, 527
 - reinforced, 530
- Conductivity, relative, for heat and electricity, 3, 624
- Cones, thickening, 171
- Conical ball mill (Hardinge), 44
- Constantan, 514, 564
- Continuous countercurrent decantation, 181, 183
- Conversion tables for electrical work, 625
- Conveyor-type driers, 10
- Copper, in metallurgical construction, 496, 499
 - electrolyte, 638
 - sampling, 78
- Copper-silicon alloys, in metallurgical construction, 561
- Corrections for optical pyrometers, 262
- Corrosion resistance, 543
- Corrosion as construction material, 552, 554
- Countercurrent decantation, 183
- Crushers, Dodge, 25
 - gyratory, 26
 - jaw, 22
 - rolls, 30
 - Symon's cone, 27
- Crushing, 20
 - general notes, 32
 - hammer mills, 28
 - Kick's law, 19
 (*See also* Grinding)
- Cuproberyllium, properties, 498
- Cupronickel, and related alloys, 513
- Current density in electrolysis, 631
- Current efficiency, 631
- Cutting down samples, 62
- Cyclone dust collectors, 398
- Cylindrical ball mills, 43

D

- Davis metal, 564, 565
- Decomposition potentials, 634
- Decomposition of sulphates, 646
- Dehne filters, 190
- Deister Concenco classifier, 124
- Deister cone baffle classifier, 125
- Deister table, 141
- Denver hydroclassifier, 120
- Denver pump, 185
- Denver thickener, 179
- Deoiling before briquetting, 410
- Depressants (flotation), 156
- Detroit rocking-electric furnace, 613
- Dewatering, C.C.D. system, 181, 183
 - costs of, 211
 - definition, 167
 - draining, stationary, 167
 - filtration, 186
 - floculation, 167
 - operating data, 209
 - sedimentation, 168
 - thickeners, 170, 172
 (*See also* Filters; Filtration)
- Diaphragm pumps, 185
- Differential-inertia dust separators, 396
- Disappearing-filament optical pyrometers, 255
- Dodge crusher, 25
- Dorr classifiers, 111-114

The subject matter of the other volume is not indexed here

Dorroco filter, 208
 Dorroco sizer, 122
 Dorroco tables, 123
 Double sintering, 320
 Draft apparatus, 440
 Drag-belt classifiers, 120
 Draining, stationary, 167
 Driers, performance of, 6, 15, 16
 Drilling copper for sampling, 78
 Drum driers, 11
 Drying, before briquetting, 410
 definition, 6
 economics, 17
 fundamentals, 6
 rapid vaporizing, 11
 Duralumin, properties, 518, 519
 Duriron as construction material, 552, 554
 Duryea metal-briquetting press, 412
 Dust collectors, cyclone, 398
 Dust-settling chambers, 395
 Dwight-Lloyd sinter, 319
 sintering machine, 308

E

Edwards roasting furnace, 291
 Electric furnace, bottom connections, 604
 classification, 594
 conductors to, 602, 603
 connections, 584, 602, 603
 construction, 620
 electronic control, 602
 resistances, 594
 series connections, 584
 Electric heat-treating furnaces, 618
 Electrical precipitators, 402
 Electrical units, 630
 Electricity, relative conductivity for
 e. and heat, 624
 Electrochemical equivalents, 631
 methods, 630
 Electrochemistry, power requirements in,
 629, 642
 (See also Electrodeposition; Electro-
 lytes)
 Electrodeposition, ampere equivalents,
 631
 anode shapes, 638
 cathode shapes, 638
 current-density effects, 631
 decomposition potentials, 634

Electrodeposition, electrochemical equiv-
 alents, 631
 electrolyte compositions, 632
 electrolyte temperature, 632
 tank arrangements, 290
 Electrodes, discussion of, 636
 Electrodes, insoluble, 636
 Electrolysis of fused salts, 641
 Electrolyte circulation, 632
 Electrolyte composition, 632
 Electrolyte temperature, 632
 Electrolytes, commercial, bismuth, 638
 cadmium, 639
 cobalt, 639
 copper, 638
 gold, 639
 lead, 639
 manganese, 639
 nickel, 639
 silver, 639
 tellurium, 639
 tin, 639
 zinc, 639

Electrolytic cell, 630

Electronic control for electric furnaces,
 602

Electroplating, 639

Electrostatic concentration, 144

Emergent-stem correction, 234

Emissivity, spectral, 260, 261

Enameled equipment in metallurgical
 construction, 570

Errors, sources of in pyrometry, 266

Evans classifier, 121

Evans-Klepetko furnace, 293

Everbrite alloy, 564

Everdur, 561

Exothermic briquettes, 428

Expansion, thermal, of refractories, 337-
 342

Fans, 441-444

 in sintering, 311

Fatigue limit, defined, 4

Feeding flotation reagents, 163, 164

FEinc filter, 206

Feld gas washer, 402

Ferroalloys in metallurgical construction,
 554, 555

Filter presses, 190

The subject matter of the other volume is not indexed here

Filters, 186
 capacities, 203
 Filtration, formulas, 187
 slime filtration, 188
 theory, 186
 Filtration-type dust separators, 399
 Fine sand concentration, 137
 Firebrick, standard sizes, 382
 Fire clay, testing, 357-359
 Fire-clay refractories, 355
 Fixed points, temperature scale, 231
 Flash roasting, 297
 Flint clays, 355
 Flocculation, 169
 Flotation, 149
 bibliography, 165, 166
 concentrate roasting, 321
 operation, 157, 159
 ores amenable to, 151
 reagents, 152-156
 Foot valves, 436
 Forsterite, 374
 Foundations, 470
 Free-settling hydraulic classifier, 121
 Frenier spiral sand pump, 437
 Frothers, 152
 Fuel briquetting, 423
 Fuels, 219
 Fume collection, 395
 Furnace-cable connections, 605
 Furnace temperatures, approximate, 384
 Fused electrolytes, 641, 642
 cell problems, 642
 raw materials, 642
 Fused silica in metallurgical construction, 573
 Fusibility of metals, 3
 Fusible alloys, 520

G

Galvanometric pyrometers, 270
 Gas, as fuel, 224
 Mond, 225
 Siemens, 225
 surface combustion, 227
 Gaseous resistors, 610
 Gases, sampling, 86
 Genter thickener, 179
 Gilmore horizontal press, 413
 Glass in metallurgical construction, 572
 Glossary of refractories industry, 385-394
 Godfrey roasting furnace, 295

Grate bars for roasters, 313
 Grate-discharge ball mill, 43
 Grinding, ball mills, 48
 closed and open circuit, 35
 controls, 53
 definition, 33
 feeders, 52
 layouts for, 54
 stamps, 35
 tube mills, 48
 Gun metal, properties, 505
 Gyratory crushers, 26

H

Hammer mills, 28
 Hand-rabbed roasting furnace, 287
 Hand sorting, 127
 Hardinge Cascade mill, 51
 Hardinge Conical mill, 44
 Hardinge countercurrent classifier, 119
 Hardinge loop-air classifier, 58
 Hardinge thickener, 179
 Hardness of metals, 3, 472
 Heap roasting, 284
 Hearth roasting, 287
 Heat, relative conductivities for, and for electricity, 624
 Heats, of combustion, 223
 specific, of refractories, 337
 Heavy-medium separation, 136
 screens in, 97
 Hegeler roasting furnace, 292
 Herreshoff roasting furnace, 293
 High-temperature thermometer calibration, 272
 Hindered-settling classification, 122
 Hérault type electric furnace, 611
 Howard dust chamber, 396
 Huntington-Heberlein process, 303, 306
 Huntington mill, 36
 Hydraulic compressors, 446

I

Ignition for roasting, 311-313
 Ignition temperatures of sulphides, 323
 Ilium, properties, 567, 568
 Impact-type dust separators, 398
 Impact-working stress, 470
 Indicating and recording thermometers, 238, 273

The subject matter of the other volume is not indexed here

Induction, furnace, 588, 594
 Induction phenomena, 591
 Induction regulator, 594
 Infraser, 147
 Insulating refractories, 375-381
 Insulators, thermal, weights of, 383
 International atomic weights, 644, 645
 International temperature scale, 230
 fixed points, 231
 Ion mobilities, 633
 Iron-chrome alloys, 555
 Iron-chrome-nickel alloys, 555
 Iron and steel classification, 472, 473

J

Jacomini briquetting press, 413
 Jaw crushers, 22
 Jigging, 129
 Jigs, continuous movable sieve power, 132
 fixed-sieve, 132
 movable-sieve hand jigs, 130

K

Kaolin refractories, 355
 Keller-Gaylord-Cole roasting furnace, 291
 Kelly filter, 189
 Kick's laws, 19
 Kirchhoff's laws, 585
 Komarek-Greaves briquetting press, 422
 Krupp furnace, 614
 Krupp mill, 42
 Kryptol (resistor), 614

L

Ladle samples, 78
 Lead in metallurgical construction, 508, 509, 561
 Lead sampling, 81
 Leggo roasting furnace, 291
 Liquids, sampling, 85
 Longmaid-Henderson process, 279

M

MacDougall roasting furnace, 292
 Magnesite refractories, 370
 Magnesium industry, briquetting in, 421
 Magnetic separation, 143
 Magnetism in metals, 4

Malleable iron in construction, 545, 546
 Malleable-iron castings, characteristics, 477
 Manganese bronze, properties, 503
 Manganese steel, characteristics of, 483
 Materials, for corrosion resistance, 543
 of metallurgical construction, 456
 bibliography, 573, 574
 strength of, 456
 at high temperatures, 544
 Mechanical concentration, 126
 Mechanical-roaster types, 295
 Mercury thermometers, 234
 errors in, 237
 Merrill filter, 191
 Merton roasting furnace, 291
 Metallic resistors, 609
 Metallics, sampling, 85
 Metallurgical fuels, 214
 Metals, brittleness, 2
 conductivity, 3
 fatigue in, 4
 fusibility, 3
 magnetism in, 4
 specific heat, 3
 strength at high temperatures, 544
 tenacity of, 3
 testing at high temperatures, 544
 thermal expansion, 544
 toughness in, 2, 471
 Metric-English equivalents, 643
 Millivoltmeters for thermocouples, 246
 Mond gas, 225
 Monel metal, properties, 513, 563, 564
 Moore "Lectromelt" furnace, 616
 Moore suction-leaf filter, 189, 192
 Motor effect of electric current, 593
 Moving loads, stresses due to, 461
 Muffle roasting furnaces, 291
 Multiblade fans, 444
 Muntz metal, properties, 503

N

National Emergency steels, 489
 Natural cements, 525
 Naval brass, properties, 500
 Nicaro Nickel Co.'s process, 278
 Nickel as construction material, 510, 562, 563
 Nickel-copper alloys as construction materials, 512, 564, 565, 566

The subject matter of the other volume is not indexed here

Nickel, electrolyte for, 639
Nickel steels as construction materials,
483, 551, 552
Nichols-Freeman flash-roasting system,
297
Nichrome, properties, 565, 567
Nip, angle of, 21
Nodulizing, rotary kiln for, 299
Nonmetallic resistors, 609

O

O'Hara roasting furnace, 290
Oil fuel, 219
Oliver-Borden thickener, 180
Oliver filter, 189, 198
Optical pyrometers, 255, 257, 259, 263
calibration, 273
window, use of, 262
Ord roasting furnace, 296
Overstrom table, 141
Oxides, spectral emissivity of, 261
Oxidizing roasting, 277

P

Panel spalling test, 343
Parks roasting furnace, 292
Pearce roasting furnace, 290, 295
Peat as fuel, 219
Pebble mills, 47
Permanent expansion of refractories, 329
Phosphor bronzes, properties, 504, 505
Pig-iron sampling, 82
Pinch effect of the electric current, 593
Pipes and piping, 432
Piston compressors and blowers, 447
Piston-and-mold briquetting methods,
418
Pistonless pumps, 433
air chambers, 436
operating, 437, 438
Pitch pockets, 538
Plant, cost estimates, 582
Plant layout, 575
Plant location, factors in, 577
size of projected, 575
smelter sites, 580
steel, location, 581
Plastic refractory, clays, 356
Plasticity of metals, 471
tests for refractory mortars, 359, 360
Plastics in construction, 573
Pneumatic separation of scrap, 410

Polarization phenomenon, 634
Polyphase alternating current, 591
Porosity of refractories, 332-335
Portland cement, 520
chemical analysis, 521
fineness, 522
insoluble residue, 521
loss on ignition, 521
magnesia, 522
miscellaneous, 524
physical requirements, 521
physical testing, 522
set, time of, 524
specific gravity, 522
sulphuric anhydride, 521
tensile set, 524
test mixture and normal consistency,
523
Potentiometers for thermocouples, 247
Potentiometric pyrometers, 269
Pot roasting, 306
Power plant air drying, 452, 454
bibliography, 455
blowers, 447, 452
compressors, 446-454
fans, 444-445
gas conveying, 438
pipes and piping, 432
requirements for electrochemical industries, 629, 642
Prerobbing for blast roasting, 306
Present value, effect of ore reserves on,
576
Pressure-leaf filters, 194
Pressure thermometers, characteristics,
238, 239
Prime movers and their choice, 429, 431
Promoters (flotation), 153
Pulsometers, 433
Pulverized fuel, 226, 227
Pumps, 432
acid pumps, 437
centrifugal pumps, 432
pistonless, 433, 436
Pyrex glass, 573
Pyrometric cone equivalent, 326
preparation, 327
Pyrometry, bibliography, 272
black-body conditions, 258, 266
calibration, 271
classes of instruments, 234
errors, sources of, 266

The subject matter of the other volume is not indexed here

- Pyrometry, high-temperature control,
 271
 indicating, 273
 optical, 255, 263
 radiation, 254, 263, 266
 recording, 268, 273
 theory, 254
 true vs. apparent temperatures, 267
- R
- Radiation pyrometers, 254, 263, 266
 calibration, 274
 Raymond separator mills, 57
 Reactance coils, 599
 Reactors, 599
 Reciprocating pumps, 432
 Reducing roasting, 278
 Refining, electrolytic, systems of, 635
 Refractories, 324
 alumina-silica diagram, 352
 basic, 370-374
 carbon, 374
 clays, firing, 353
 estimating brickwork, 383
 fire clay, 355
 fire clay, testing, 358
 forsterite, 374
 furnace temperatures, 385
 glossary, 385-394
 insulating, 375-381
 kaolin, 355
 materials (general list), 392-396
 mechanical strength, 350
 mineralogical composition, 325
 panel spalling test, 343
 permanent expansion, 329
 plastic clays, 356
 plasticity testing, 359
 porosity, 332-334
 pyrometric-cone equivalent, 326
 rocks, list of, 392-396
 semifint clays, 356
 silica-alumina series, 351
 silica refractories, 361, 364-365
 siliceous clays, 356, 366
 spalling test, 343
 specific heats, 337
 spinel, 375
 superduty fire-clay brick, 368
 test cone preparation, 327
 testing fire clays, 358
- Refractories, thermal properties, 337-341
 types of, 351
 weights of, 383
 zirconia, 374
 Refractoriness, 326
 Rennerfelt furnace, 616
 Repeated and reversed stresses, 472
 Resistance of refractories, to compression,
 336
 to slagging action, 350
 to spalling, 341
 Resistance furnaces, 594
 Resistance pyrometry, 230
 Resistivities at high temperature, 621,
 622
 Resistivity of metals, 623
 Resistor connections, 603
 Resistors, nonmetallic, 609
 Retaining walls, 470
 Rezistal steels, properties, 550, 551
 Richards hindered-settling, classifier, 124
 Richards-Jannev classifiers, 124
 Richards launder-vortex classifier, 121
 Richards pulsator classifier, 125
 Ring roll mills, 37
 Rittenger's law, 19
 Riveted construction, 491
 Roasting, 278
 blende, 283
 chemical requirements, 281
 chloridation, 279
 chloridizing, 279
 circular furnace for, 292
 double, 320
 flash, 279, 298
 flotation concentrates, 321
 heap, 284
 mechanical requirements, 283
 methods, 280
 oxidizing, 277
 practice, 280
 prerosting, 306
 reducing, 278
 in stalls, 284
 sulphating roasting, 279, 646
 thermal requirements, 283
 Roasting and sintering, automatic pro-
 portioning, 314
 Rod mills, 49
 Roller grinding mills, 39
 Rolls, crushing, 30
 fine grinding, 37

The subject matter of the other volume is not indexed here

Ronay-Gilmore briquetting press, 411
 Ropp furnace, 290
 Rotary driers, 8
 Rotary-hopper dewaterer, 209
 Rotary kilns, 299
 Rotary pumps, 437
 Rotary vacuum filter, 198
 Rotating-drum grinding mills, 42

S

Salting samples, 86
 Sampling, 60
 bibliography, 87
 coal, 66
 contamination in, 70, 71, 72
 of copper, 78
 crushing for, 65
 cutting down, 62

 ladle samples, 78
 lead, 81
 of liquids, 85
 mechanical, 64
 metallics, 85
 pig iron, 65
 size-weight ratios in, 66
 Sand pump, Frenier spiral, 437
 Sand tables, 138
 Savelsburg process, 304, 306
 Screening, 89
 machines, 90, 93
 screen testing, 100
 Screens, for closed-circuit grinding, 97
 in heavy-media separation, 97
 standard test, 99
 testing, 100
 Scrubber-dust separators, 401
 Sedimentation, 168
 Semiflint refractory clay, 356
 Separation by sliding friction, 147
 Setting up pumps, 438
 Settling chambers (dust), 395
 Settling ratios, classification, 105
 Settling velocities, quartz and galena, 128
 Sharples centrifugal separator, 213
 Shatter test for coke, 217
 Sheet brass in construction, 504, 505
 Sheet copper in construction, 499
 Shriver thickener, 181
 Sieves, standard, 99
 Silica-alumina series (refractories), 351
 Silica refractories, 361, 364-365

Siliceous-clay refractories, 356, 366
 Silico-manganese steels, 489
 Silicon irons in construction, 552
 Sintering, by blast roasting, 301
 fans in, 311
 Sintering practice, 316
 Size-weight ratio in sampling, 66
 Sizing (screening), 89
 Skinner roasting furnace, 294
 Slag resistor, 609
 Slime tables, 142
 Slime treatment, 143
 Sly dust arrester, 401
 Snyder sampler, 65
 Söderberg electrodes, 608
 Solders, 507, 508
 Spalling test for refractories, 343
 Specific gravity tests on refractories, 335
 Spectral emissivity of materials (table), 260
 of oxides, (table), 261
 Spigot type classifier, 107
 Spinel refractories, 375
 Spirlot roasting furnace, 295
 Stall roasting, 284
 Stamps, 35
 Standard screen scale sieves, 99
 Steam-jet blowers, 446
 Steam siphons, 437
 Steel buildings, construction, 495
 Steel castings in construction, 480
 Steel members (construction), 492
 Steel pipe specifications, 481
 Stevenson-Little briquetting, press, 413
 Stone in construction, 534
 Storage for pulverized fuel, 226
 Straight-line roasting furnaces, 290
 Stress, special problems in, 463
 Structural carbon, 573
 Structural-steel design, 489, 490
 specifications, 481
 Structural theory, 456, 464
 Suction lift, 437
 Sulphates, equilibrium data for, 646
 Sulphating roasting, 278, 279, 646
 Superduty fire-clay brick, 368
 Superpanner, 147
 Surface combustion, 227
 Surface-current classifiers, 107
 Sweetland filter, 181, 196
 Symons cone crusher, 27
 Systems of loading (construction), 457

The subject matter of the other volume is not indexed here

- Tank arrangements, 635
 - Tanks, construction of, 470
 - Tellurium, electrolyte for, 639
 - Temperature-e.m.f. relations in thermocouples, 245-246
 - Temperature resistance of construction materials, 543
 - Temperature scales, 230
 - fixed points on, 231
 - Template, drilling by, 73
 - Tenacity of metals, 3
 - Test cones, preparation of, 327
 - Thermal expansion of corrosion resistants, 572
 - of metals, 544
 - of refractories, 342
 - Thermocouples, 252-272
 - calibration, 272
 - common return, 252
 - comparison methods, 272
 - fixed installations, 273
 - installations, 251
 - instruments, 245
 - millivoltmeter, 246
 - potentiometer, 247
 - Thermometers, 234
 - automatic temperature control, 270
 - emergent-stem correction, 234
 - mercury, 234
 - partial immersion, 238
 - pressure, 239
 - recording, 268
 - thermoelectric, 240
 - tolerances, 235, 236
 - Thermometry, bibliography, 274
 - classification of thermometers, 234
 - resistance, 268
 - theory, 254
 - (See also Pyrometry)
 - Thickeners, c.c.d. cyanide system, 183
 - multiple-compartment, 174
 - Thickening, filtration, 179
 - Thiessen gas washer, 402
 - Timber, 538
 - classification, 538
 - defects, 538
 - general, 538
 - pitch pockets, 538
 - preservation, 542
 - specifications, 539, 540
 - Timber, wane, 538
 - Timken briquetting press, 413
 - Tin as construction material, 507
 - Tin, electrolyte for, 639
 - Tin pipe, 507
 - Tobin bronze properties, 502
 - Top-charge electric furnaces, 617
 - Torsion, 462
 - Toughness of metals, 2, 471
 - Trail system of flash roasting, 298
 - Transformation-point indicators and recorders, 270
 - Transformers, electric furnace, 598
 - Transverse bents, 467
 - Trent process for fuel, 228
 - Trusses, 465
 - Tube mills, 48
- V
- Vacuum chambers on pumps, 436
 - Vanners, 142
 - Vezin sampler, 65
- W
- Wane, definition, 538
 - Washers, preliminary, 127
 - Waste-heat boilers, 430
 - Water of plasticity, testing, 359
 - Wedge roasting furnace, 294
 - Weeks electric furnace, 616
 - Weights, atomic, table of, 644, 645
 - Wet-surface impact separator, 399
 - Wethey roasting furnace, 290
 - White metals, properties, 510, 511
 - Wile furnace, 614
 - Wilfley tables, 139
 - Window, use of, in optical pyrometry, 262
 - Wire-hung dust chamber, 396
 - Wood as fuel, 219
 - Wrought iron, properties, 477, 545, 546
 - Wrought steel, properties, 545, 546
- Y
- Young's modulus, definition, 3
- Z
- Zellweger furnace, 291
 - Zenith filter, 189
 - Zinc in construction, 506
 - Zinc, electrolyte for, 639
 - Zinc roofing, 507
 - Zirconia refractories, 374
 - Zone box for thermocouples, 253

The subject matter of the other volume is not indexed here

